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A Study of the Blast Furnace

By

Harbison-Walker Refractories Co.
Pittsburgh, Pa.

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CONTENTS

INTRODUCTORY 9

Map in colors showing Blast Furnace Districts and Iron Ore Deposits of the United States attached to inner back cover.

CHAPTER I.— DESCRIPTION OF PLANT.

PAGE

GENERAL SKETCH 11
FURNACE — Hearth or Crucible 14
Bottom 14
Tapping Hole 16
Cinder Notch 16
Tuyeres 18
Tuyere Stocks 21
Bustle Pipe 21
Hot Blast Main 21
Bosh Construction and Lining 22
Bosh Cooler Plates 23
Mantle 24
Inwalls 24
Top 25
Filling Device 26
Stock Distributors 26
Openings 27
Piping 29
Downcomer 29
Dustcatcher 30
Gas Mains 30
Boiler Plant 31
Blowing Engines 31
Cold Blast Main 32
HOT BLAST STOVES — General Description . . 32
Valves 37
DISPOSAL EQUIPMENT 52
Pig Beds 52
Chills 54
Casting Machine 54
Slag Disposal 55

33X708

CHAPTER II.— MATERIALS — ORES. PAGE

ORES

Hematites 58
Brown Ores 61
Magnetites 62
Carbonates 64
Summary of Production 65
Items Affecting Valuation 68

TREATMENT OF ORES.

Roasting 71
Calcining 75
Concentration 77
Nodulizing 80
Briquetting 82

CHAPTER III.— MATERIALS — FLUXES AND FUELS.

FLUXES — Composition 84
Relative Value 87

FUELS

Coke 92
Impurities 93
Manufacture—Beehive Oven . . . 98
Mitchell or Belgian Oven 101
By-product Oven . . 102
Charcoal 104
Anthracite Coal 105
Valuation of Fuel 106

CHAPTER IV.— BURDENING THE FURNACE.

INTRODUCTORY 108
SLAG 109
Composition 112
Control 113
DETAIL OF PROBLEM IN BURDENING . . 116
Flux 117
Fuel 117
Ore 118
Fuel Requirements 119

CHAPTER V.— OPERATING THE FURNACE. PAGE

Blowing in 122
Charging 125
Operation 126
Banking the Furnace 129
Blowing out 131
Operation of Stoves 131
Furnace Troubles 134
Destruction of Lining 134
Break-outs 138
Loss of Tuyeres and Cooler Plates . . . 138
Pillaring 139
Scaffolding 140
Wedging 141
Chilling 142

CHAPTER VI.— FURNACE REACTIONS.
GENERAL DISCUSSION 144
Flux 147
Fuel 150
Ore 150
Cyanides 157
Carbon Ratio 158
Dry-Air Blast 159

INTRODUCTORY

IN the effort to work along the lines making for maximum efficiency, we have long realized the benefit of having the members of our organization as familiar as possible with at least the underlying principles concerning the various industries in which fire brick and other refractories form so vital a part of the equipment. "A Study of the Blast Furnace" was prepared with this end in view, and is merely a digest in as simple and brief a form as possible, of the widely scattered information relating to blast furnace operation that has appeared in the technical press.

Of necessity, there is little of original investigation contained in this volume. All available information has been most freely drawn upon, particular indebtedness being acknowledged to the government reports on "The Production of Iron Ores," to Forsythe's work, "The Blast

Furnace and the Manufacture of Pig Iron," and to Campbell's volume, "The Manufacture and Properties of Iron and Steel."

As mentioned in the preface of "A Study of the Open Hearth," the present volume forms the second of the series.

DESCRIPTION of PLANT

CHAPTER I

FURNACE

GENERAL SKETCH — The blast furnace, in which is conducted the manufacture of pig iron, is merely a cylindrical steel shell lined throughout with fire brick. This shell varies in height from 40 to 100 feet or even higher, and in each furnace has varying diameters from top to bottom, the lines of the furnace being thus adjusted to the various changes going on within it. The walls of the hearth near the bottom of the furnace are pierced with openings through which so-called tuyeres supply a strong blast of heated air to unite with the carbon of the fuel.

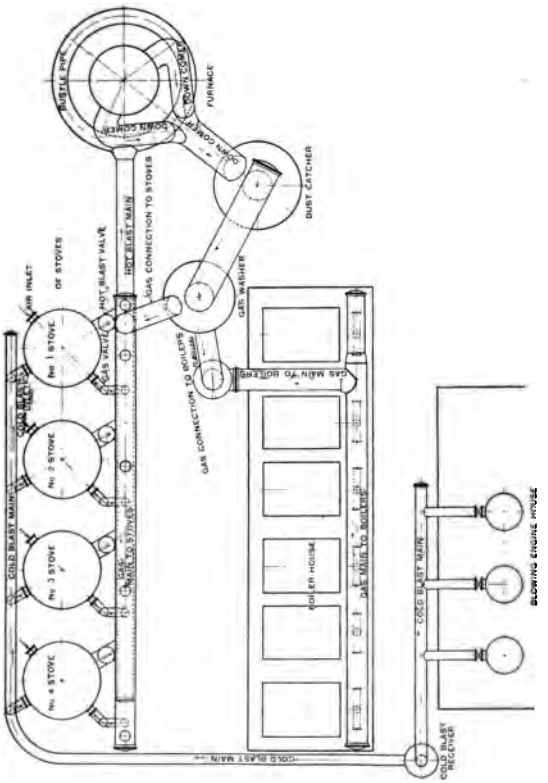
Into the furnace top is charged continuously the ore, fuel and flux which go to make up the burden.

The ore furnishes the iron for which the furnace is operated; the fuel in combustion gives off gases which serve to reduce the iron

to a metallic form, and also supplies the heat necessary for the reactions which occur within the furnace, and to melt the resultant products. The flux serves to unite with various compounds which would otherwise be infusible at furnace temperatures and so, not only removes in a fluid state the ash of the fuel but the earthy materials and impurities occurring in the ore. It also serves in such combination as the means of controlling the amounts of certain elements desirable in the iron, but desirable only within limited percentages.

As the charge slowly works its way downward, approaching the zone of highest temperature at or slightly above the tuyeres, the various reactions become more and more complete and, finally, fusion of the resultant products occurs, the molten material collecting in the hearth of the furnace, which serves as a reservoir. The molten iron being of greater specific gravity than the impurities, sinks to the bottom while the impurities of the ore and ash, together with the flux, combine to form a slag which floats on the surface of the iron. The two can then be easily tapped off separately through openings located at proper levels.

The gaseous products rising through the descending column of ore, flux and fuel, pass off through openings at the top and being combustible, are led through the downcomers to the hot blast stoves and to the boilers where they are burned.



DIAGRAMMATIC SKETCH SHOWING PIPING FOR FURNACE AND STOVES

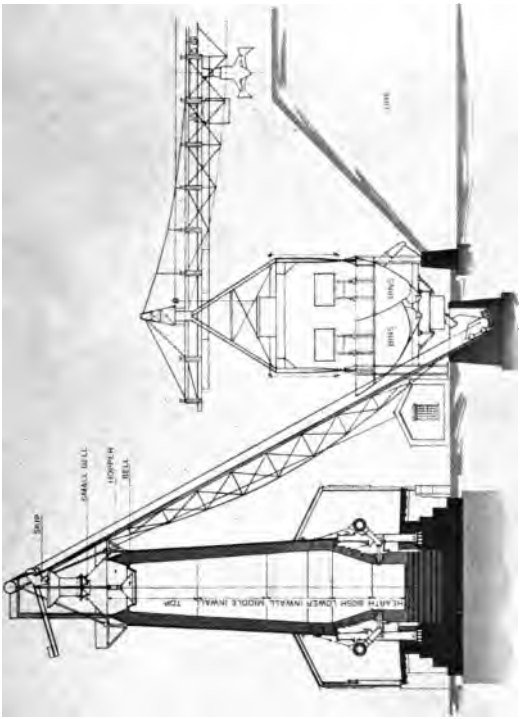
The diagrammatic sketch shown on page 13 will illustrate the passage of the gases and the order in which the general construction will be considered.

The furnace structure, a representative type of which is shown on page 15, will be considered under the three main divisions of:

Hearth or Crucible,
Bosh,
Inwalls. { Lower Inwall / Middle Inwall / Upper Inwall or Top

HEARTH OR CRUCIBLE — This portion of the furnace serves as a receptacle for the molten slag and iron. Its fire brick lining is from 27″ to 31½″ in thickness or even thicker, dependent on the size of the furnace. It is surrounded by a steel jacket either of heavy riveted plate or still heavier castings of steel or iron in segmental form. In the first instance, the plate is often cooled by a direct water spray, while in the second, the castings are sometimes made sufficiently heavy to require no cooling, or if cooled, this is done by a trough of water running around the exterior. In a number of furnaces, pipes for cooling water are cast in the interior of the metal composing the hearth jacket itself. Page 17 shows a section of the hearth and bosh of a typical furnace.

Bottom — The bottom is composed of a solid mass of fire brick, either in the standard sizes or in heavy blocks. The thickness varies

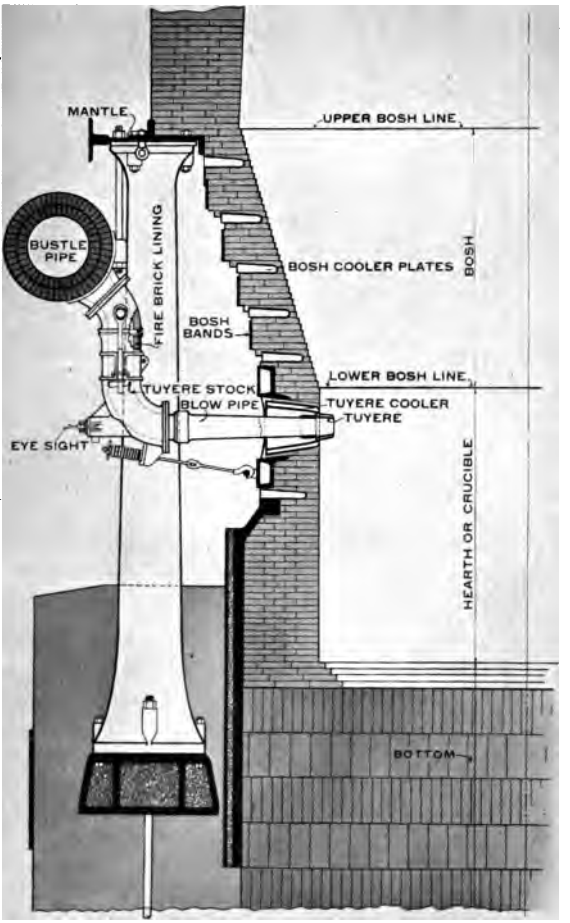


SECTION OF BLAST FURNACE SHOWING FILLING ARRANGEMENT, BINS AND ORE BRIDGE

with the size of the furnace, from 5′ to 6′ in the case of the smaller furnaces, to 12′ or 14′ in the larger furnaces, and must be very closely laid. If this is not done or if the joints are not properly broken, the molten metal tends to seep through the cracks and an enormous amount may so escape from the hearth. The high grade fire brick are underlaid by fire brick of a cheaper grade and this in turn by concrete in a deep and broad foundation, giving absolute stability.

Tapping Hole — This is the opening through which the iron is drawn off and is usually located from 18″ to 2′ above the furnace bottom, and at the front of the furnace or facing the pig bed or cast house. It is merely an opening several inches square left in the brickwork and is not usually water-cooled, as in case there should be any leakage of water disastrous explosions would be sure to result on contact of the water with the iron. Except when the furnace is being tapped, the opening is closed with a semi-plastic clay mix which soon burns into place so that it is as firm as the hearth walls themselves.

Cinder Notch — There is usually but one cinder notch, although there may be two or three in the larger furnaces. As its name implies, this opening is for the removal of slag or cinder and is placed at from 45° to 90° from the tapping hole. Since cinder only is to be

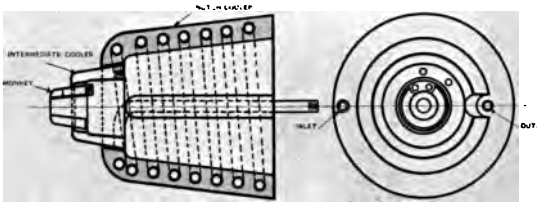


SECTIONAL VIEW
BLAST FURNACE HEARTH AND BOSH

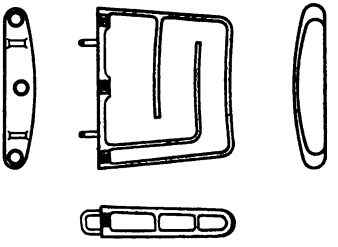
drawn from it, it is evident that its height determines how high the iron may reach before tapping; in other words, with any given furnace diameter, the cinder notch is so placed as to allow accumulation of the desired quantity of metal before tapping. It is water-cooled, being made up of a series of hollow tapered bronze, copper, or cast-iron coolers in the shape of a frustum of a cone. Page 19 shows the several parts, the outer and largest being usually termed the cooler, the next smaller the intermediate cooler, and the innermost, reducing the diameter to 2″ or 2½″, is called the monkey. This small opening is stopped by a tapered iron plug attached to an iron bar by which it may be withdrawn when it is necessary to flush off the slag.

To each cooler are connected two water pipes for ingress and egress, thus allowing a good circulation and consequent cooling. Should the iron be allowed to rise high enough to reach these coolers, it would rapidly cut them out, as such is the effect of molten iron on copper or bronze. On the other hand, the slag while not affecting the bronze, would very rapidly cut away the brickwork, were no cooler in place. Molten iron itself has comparatively little effect on the fire brick.

Tuyeres — The tuyeres through which the blast is admitted are symmetrically placed around the circumference of the hearth and



CINDER NOTCH



SLAG COOLER PLATE

determine the height to which the slag may be allowed to rise. This may be anywhere from 2½′ to 4′ above the cinder notch. The number of tuyeres varies widely with the ideas of the designer and the size and practice of the particular furnace. There are usually from 10 to 16. The tuyere with its cooler is shown on page 17.

The outer ring is termed the cooler and is practically identical with the cooler on the cinder notch except that it is flush with the interior and usually with the exterior wall. Into it fits the tuyere proper, projecting somewhat into the furnace. It also is hollow and cooled by water circulation in the same way. The size of the tuyere opening is important because of the direct relation it bears to penetration, pressure and volume of blast; it varies roughly from 3½″ to 7″ in diameter.

Against this inner tuyere rests a horizontal cast-iron pipe termed the blow-pipe, connecting in its turn with a pipe called the down-leg, tuyere stock, or boot-leg, which takes off from the bustle pipe encircling the furnace and supplying its blast. The blow-pipe is not lined and merely fits into place with a ball and socket joint, ground at each end, and is supported by the pressure of the tuyere stock, the latter swiveled and attached to the furnace bands by a strong spring.

Tuyere Stock — As indicated on page 18 this carries the blast from the bustle pipe to the blow-pipe. It is usually in two sections of cast iron and has a 2″ to 3″ lining of fire brick.

Directly in line with the blow-pipe is a perforation or eye-sight in the back of the tuyere stock. This opening is closed by a piece of glass or mica so that an inspection of conditions in the hearth can easily be made and also a bar be thrust through, if necessary, from tuyere to tuyere, to dislodge any slag or other obstruction which may be closing the tuyere opening.

Bustle Pipe — This is a cylindrical sheet steel pipe lined with from 9″ to 12″ of fire brick, and encircles the furnace about 10′ to 15′ above the floor. It is usually suspended by brackets or straps from the furnace columns. Its diameter is proportioned to the volume of blast it is to carry.

Hot Blast Main — This is a fire brick lined pipe carrying the hot blast directly from the stoves and terminates in the bustle pipe. Its lining is from 9″ to 12″ in thickness, and the selection of a brick suitable for the conditions encountered here and in the bustle pipe and tuyere stocks is most important. On the "hot side" of the furnace, as this system of piping is termed, the lining is subjected to high as well as varying temperatures and must be hard and tough to resist the scouring effect

of the blast often laden with dust as it comes from the stoves. On the other hand, it must not be so dense as to spawl and crack from sudden changes in temperature and must at all times possess ample refractoriness.

BOSH — *Lining* — This portion of the furnace, see page 17, is that extending from the upper line of the hearth about to the mantle or to the lowest portion of the inwall. It is in the form of an inverted frustum of a cone, its limits being termed the upper and lower bosh lines respectively, and is of one of two types of construction. The more usual is that shown on page 17 and practically all the larger furnaces are so constructed. No shell proper covers this but the brickwork is built up to the desired lines and is then supported by numerous very heavy steel bands completely encircling the bosh. In the brickwork, 27″ or more in thickness, there are inserted every few courses between the bands above referred to, bronze or copper cooling plates with a circulation of water.

In the other type of bosh construction, no interior cooling plates are used, and but a thin layer of fire brick, say from 9″ to 13½″. Around this is a continuous steel shell cooled by troughs of water, the water flowing down from one trough to that below, or the trough may be riveted on in spiral shape, thus merely allowing the water to circulate around the shell.

However, boshes lined in any other way than with interior cooling plates are becoming fewer.

The bosh as a whole is the zone of fusion, and the stock, as it approaches this zone, becoming more and more pasty, occupies less and less space and finally melts. The narrowing lines of the bosh thus aid in concentrating the softening and semi-molten mass before the tuyeres, as well as afford it support.

Bosh Plates— There are a number of styles of plates differing, however, but comparatively little, such as the Scott, Gayley and Kennedy plate. All are of bronze or copper, iron being a poorer conductor of heat, and melting or burning away much more readily. A typical plate is shown on page 19. In general, in the Scott plate, the water entering the plate is led directly to the nose, finding its way back through a series of baffles; the Gayley has but one water-way, and that but 10″ to 12″ in depth from the nose, while the Kennedy is similar to the Gayley, but with two water-ways. The plates are usually of the same length as the thickness of the wall, but in order to save somewhat on the original cost, they are sometimes made but half this length, the remaining thickness of the opening in the wall being taken up by a cast-iron box open at each end through which this short plate slips to the interior wall of the furnace. This construction also makes somewhat easier the renewal of any plate, should it be burned out.

Mantle -- Just above the upper limits of the bosh is the mantle. This is circular in form, usually built up of heavy steel plates and angles and on it rests not only the entire weight of the furnace shell proper but the immense mass of brick lining beside. This mantle in turn is supported by a series of columns either of cast-iron or built up of structural shapes, the foundations of these columns going down below the hearth level often to the furnace foundation itself. Such construction allows the entire hearth and bosh of the furnace to be torn down without disturbing any of the work above the mantle.

INWALLS — The inwalls comprise those portions of the furnace from the mantle up, and in turn are often considered as being divided into the lower inwall, say the first 25′, the middle inwall the next 20′ to 25′, and the upper inwall, or top. This entire portion of the furnace is surrounded by the usual riveted steel shell, the lining being from 27″ in thickness to even 60″ at the mantle, as is the case in some of the large furnaces. From here, the thickness of the walls tapers somewhat toward the top.

From the mantle up for varying distances there are often cooler plates embedded in the furnace walls. This practice is unquestionably extending and the tendency is to carry the plates higher and higher. These in a measure serve to protect the brick from cutting out under the severe conditions of modern driving.

At the top of the furnace, the bell and hopper, i.e., the charging device, close the opening, and for some distance below this, the portion termed the stock line, the walls may be vertical. Often the slight outward flare of the furnace lines is continuous from the top entirely to the upper bosh line, this flare allowing freedom of movement for the descending charge.

TOPS — The blast furnace of the earliest type was left entirely open at the top, allowing the gases to burn freely, but there was soon a realization of the heat wasted and the top was then closed with a bell and hopper, as on page 15. The hopper forms merely an annular V-shaped bin at the top of the furnace and holds the accumulation of the charge until the proper time to dump it.

The bottom of this hopper or bin is formed by the bell which effectually seals the opening until such time as the bell is lowered to discharge its contents into the furnace, but has the defect of allowing the escape of gases during that period.

To overcome this, a second smaller bell was introduced, as on page 15. The skip dumps first upon this smaller bell which, in turn, lowers to allow the charge to slide to the larger bell, from which it discharges into the furnace. The whole arrangement being enclosed in one casing, there is no opportunity for

the escape of gases. The bell is of very heavy cast steel or iron, with the sides at an angle of about 45° so that the charge may easily slide off, and is lowered sufficiently to clear the engaging ring by some 18″ to 24″.

Filling Devices — In the old type of furnace where the tonnage was small, the furnace was hand-filled, i.e., the charges of fuel, flux and ore were wheeled in buggies from the bins to a vertical elevator beside the furnace stack.

The buggies being hauled to the top, the top fillers emptied them, returning them to be refilled. With the advent, however, of the modern furnace of large capacity, it became more difficult to supply the fuel, flux and ore fast enough to take care of the furnace requirements.

Furnaces are now designed with an automatic skip arrangement whereby the skip is hauled up an inclined skipway extending from the storage bins at the base to the furnace top and there automatically dumped. This arrangement, however, has introduced a number of complications which will be discussed later.

Stock Distributors — As previously mentioned, the advent of the automatic skip-filling arrangement introduced difficulties not at first recognized but which have been very widespread. It is apparent that as the skip dumps into the hopper of the receiving bell the coarsest

lumps of ore, stone, etc., will roll faster than the fines, and accordingly tend to gather at the side opposite the skip. The same tendency holds good as the ore is dropped to the stock in the furnace. The result is that there is a preponderance of the coarser ore on the side of the furnace opposite the skip and, consequently, less resistance to the passage of the gases, resulting in an increase in the volume of gas passing up this side of the furnace. This produces greater heats and results in a tendency to erode and cut away the lining, creating a so-called "hot spot" which necessitates a water spray on the outside of the shell to cool it. Various devices have been designed to overcome this, such as the Brown Rotary Distributor, the Baker-Neuman, the Roe Distributor, the Kennedy Top, the McKee Top, the Neeland Top, and others.

These devices vary widely in the details of their construction and the principles on which they operate. Some rotate through a certain arc, thus dropping the charge at different segments of the circle; others use a cylindrical skip the bottom of which drops, allowing the charge to fall vertically upon the center of the bell, while others have arrangements of baffle plates to deflect the stock as desired. However, details of the many arrangements can hardly be discussed here.

Openings — For the escape of the gases *incident* to the operation of the furnace, from

BAER EXPLOSION VALVE

one to four openings are provided, piercing the furnace top, and from these openings lead fire brick lined pipes termed the downcomer, or downtake, which in turn lead to the dust catcher. There are also often provided other openings closed by weighted doors, called explosion doors, which in case of a slip, give relief to the temporarily high pressure of gases. This construction is not generally used in the newer furnaces, where the top is made sufficiently heavy to withstand this excess pressure.

The Baer Explosion Valve is said to have given excellent results. This type of valve may be set to raise at any desired pressure, affording gradual release. Where installed it has entirely overcome the troubles due to ejection of stock from the furnace top.

From the top of the downcomer extend short vertical pipes called bleeders, which are closed with a valve at the top and thus allow the escape of gas into the air at any time this may be necessary. These are usually lined with from 2½″ to 3″ of fire brick.

PIPING — *Downcomer* — The diameter of the downcomer is dependent on the size of the furnace and volume of gas given off. It should be of such size, however, as to prevent a too rapid flow of gases, which occur with excessive top pressures. Such high pressures result in carrying an excess of dust and consequent wear on the brickwork, especially with the

modern tendency toward harder driving. It usually has a 9″ fire brick lining.

Dust Catcher — The object of the dust catcher, as its name implies, is to reduce as much as possible the amount of dust which shall be carried beyond it, this dust causing much trouble from clogging of the flues, stoves, boilers, etc. The principle of the operation of the dust catcher is merely that of the decreased velocity due to increasing the area of the conducting pipe, thus allowing the dust to settle. For this reason the greater the difference between the diameters of the downcomer and the dust catcher, the greater the percentage of dust which will be deposited.

The dust catcher is a large brick-lined vertical steel cylinder from 15′ to 30′ in diameter, usually with a dome or conical top and with an inverted conical bottom which has an opening through which the accumulated dust can be discharged. With the ordinary type of dust catcher, the flue dust in the gas can usually be reduced sufficiently to give satisfactory results in the stoves and under the boilers, but if the gas is to be used for gas engines, it must be subjected to further treatment in some type of gas washer. There are a number of these, including both stationary and centrifugal washers.

Gas Mains — From the dust catcher or gas washers, as the case may be, the gas is piped

through the main directly to the stoves and to the boilers or gas engines, for fuel. This pipe usually has a 4½" fire brick lining.

The lining for the "cold side," as it is termed, i.e., the downcomer, dust catcher and gas main, is naturally not usually subject to as high temperatures as is the hot side, but there are times in nearly every furnace campaign when the top temperatures and consequently the temperatures in the cold side piping rise several hundred degrees above normal and it is merely the part of wisdom to be fully insured against trouble from this source by a brick lining amply refractory.

In addition to the effect of the heat the brick are continuously subject to the rush of furnace gases laden with dust from the ore, coke and limestone and the abrasion is oftentimes very severe. An intelligent selection of suitable clays gives a brick very refractory but also dense and tough.

Boiler Plant — The boiler plant to supply power for the blowing engines, lighting, pumping, hoisting, etc., differs in no important particulars from the ordinary power plant, save that it is usually so arranged as to operate with furnace gas as a fuel. It must also be adapted to use coal, should the furnace be out of blast, or not working properly, and its normal fuel supply thus be unavailable.

Blowing Engines — These engines supplying the blast for the furnace obviously form a

most important portion of the furnace equipment but one which can hardly be considered in detail here. The engines are usually compound and are either vertical, horizontal or a combination of the two. The turbine blower is also being introduced. While until recently steam was in this country universally the motive power, a number of the largest and most recent installations have been gas engines.

The one thing which has retarded the earlier adoption of the gas engine operating with blast furnace gases has been the difficulty of economically and thoroughly washing the gases so as to give good and reliable results. This now seems to have been accomplished and in all probability future construction will see an increasingly large proportion of gas engine driven installations.

Cold Blast Main — This is the main which conducts the blast from the engine to the stoves.

HOT BLAST STOVES

GENERAL DESCRIPTION — These stoves, so called, are for the heating of the blast previous to its introduction into the furnace, the heat being derived from the burning of the gases given off from the furnace during its operation.

The first stoves in use were of cast-iron. The gases were burned around and circulated among U-shaped cast-iron pipes enclosed in a fire brick structure; the blast in turn passing

through these tubes, picked up considerable heat. This process was continuous — a recuperative process. However, it was subject to a number of defects among which was the burning out of the tubes, making it impossible to obtain more than 900° F. in the blast. This type of stove was followed by the fire brick stove operated on the regenerative principle and by its use a hot blast temperature of approximately 1500° F. can be obtained.

The outside steel shell, some 18′ to 22′ in diameter and 80′ to 100′, and in some cases even 130′ high, encloses a structure of fire brick which consists essentially of two parts, i.e., the combustion chamber and the checker work. This combustion chamber is an open vertical shaft extending from the bottom to the top of the stove. The gas is introduced at the bottom, together with a suitable supply of air through air valves, to give proper combustion. The gas burns, developing a high temperature, and the escaping products of combustion passing up the combustion chamber and then down through this series of regular parallel flues called "checker work," give off the major portion of their heat to the brickwork and then escape through the stack or chimney. The checker work may be so divided as to have one, two, three or even more passes before the gases escape to the outer air,

the number of these dependent on the type of stove.

The mass of brickwork having attained a high temperature, the valves are reversed and the blast from the engine is driven through the stoves in a reversed direction, and in passing over the heated brickwork, is raised to possibly 1300° or 1400° F. This blast is kept on until the temperature, naturally falling as the stove cools, is reduced to 1100° or 1200° F. when the stove is put on gas and allowed to store up a further supply of heat. However, it must not be inferred that the temperature of the blast as it issues from the tuyeres is normally allowed to vary by any such amount. Such a variation would have too great an effect in changing the height of the zone of fusion and would result in very irregular running.

The usual method is to keep the temperature of the blast proper as nearly constant as possible. This is effected by opening the valve in the mixer pipe, as it is called, allowing cold air to mingle with the air from the stove, then, as the stove gradually cools, the valve is more nearly closed and the blast is finally at the temperature attained from the checkers of the stoves.

In this way the air from a stove may be at, say 1300° F. when first put on the furnace, the mixer valve opened sufficiently that the

temperature of the air at the tuyeres be only 1100° F. but this latter temperature will be maintained for a full hour in spite of the drop in the stove, by the gradual changing of the valve in the mixer and consequent decrease in the amount of cold air admitted.

By this method of operating, a practically constant temperature is maintained. If a stove is comparatively new and in good condition, it should hold its temperature with a drop of not more than 100° to 175° F. in an hour. If the brick have become vitrified, however, or the stoves are dirty, it will not hold up as well.

It is usual to have four stoves, so that while one is in blast the other three may be on gas. With such arrangement it is customary to have a stove in blast about one hour, while with four stoves, it has three hours in which to become hot. In some of the newer installations five stoves are installed, so that the full complement may be available while the cleaning of one stove is in progress.

There are few important locations where fire brick are used that have had less real study and intelligent consideration by the user than in the hot blast stove.

Those men who are giving the matter most thought and on whom rests the burden of responsibility for economical furnace operation,

are advocating the use of the highest grade stove brick obtainable. The use of first quality brick, not only through the first pass and well into the second pass of stoves, but even to within a short distance of the stove bottom, is advocated by many.

The lack of proper attention to the importance of quality in stove brick has undoubtedly been due to insufficient data regarding the actual performance of various qualities of brick in the stoves. Furthermore, there is the usual temptation to accept the very considerable saving in first cost in so large an item as fire brick for large stove installations when the consequences of such a decision can be put forward into that indefinite but very positive future when all basic shortcomings in construction become apparent.

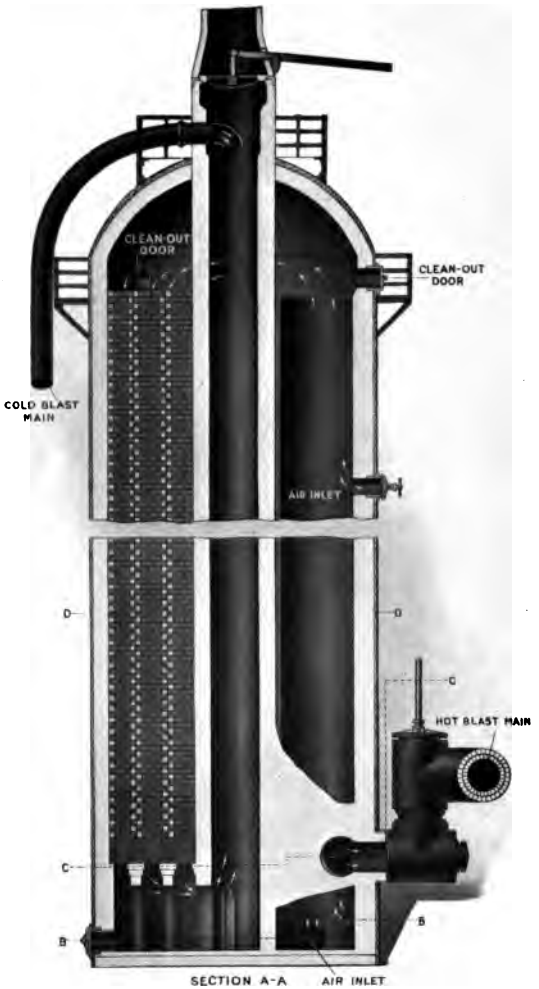
In the selection of a brick, what is of vital importance is not what is paid per thousand brick but what is actually paid per thousand heat units. The office of the brick is to take on and give off heat units and how well it performs this test depends strictly upon the maintenance of its thermal capacity. Two brick may show up equally well in a straight heat test, yet the one, after a few months of hard usage, becomes vitrified, its thermal capacity lowered, while the other is practically unaffected and will continue to perform its work for years. It is safe to say that the day

is coming when the factor of efficiency in stove refractories will be appreciated more nearly at its worth and the blast furnace manager will not permit himself to be handicapped by inefficient equipment.

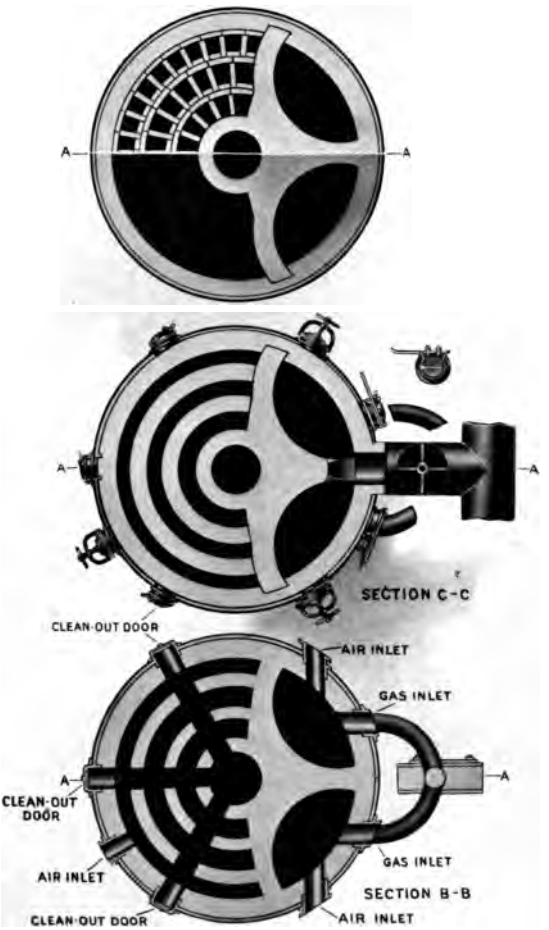
In general, stoves are divided into two classes, the side combustion and centre combustion, types of which are shown on pages 38 to 51, inclusive.

There may be a stack for each stove, or one only for the whole group. This is sometimes placed at the end of the line of stoves, but when so placed, tends naturally to favor the stoves nearest it, and for this reason is more often about midway in the group. In certain types of stoves, such for instance as that shown on page 42, a chimney is provided for each at the top of the stoves. Chimneys or stacks are all provided with fire brick linings.

VALVES — The valves to be considered consist of the cold blast valve, hot blast valve, air valve, gas valve and chimney valve. The cold blast and air valves admit air only at atmospheric temperature or slightly above it as the air comes from the engine. Some ordinary form of gate valve is here satisfactory. The gas valve admits gas at furnace top temperatures, i.e., 400° to 600° F. except as it is cooled by radiation. It is not usually water-cooled. The temperature to which the chimney valve is subjected is dependent upon



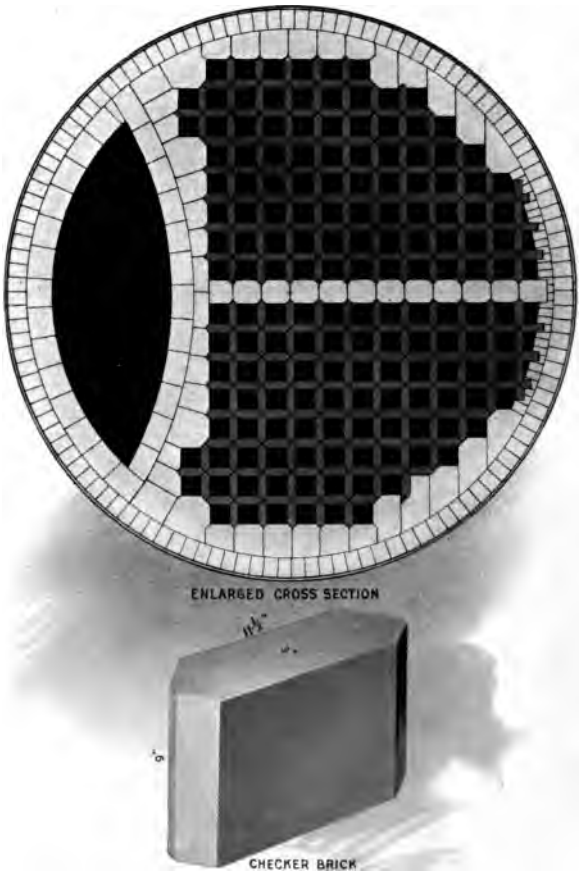
CALDER HOT BLAST STOVE
THE S. R. SMYTHE CO., Engineers, PITTSBURGH, PA.



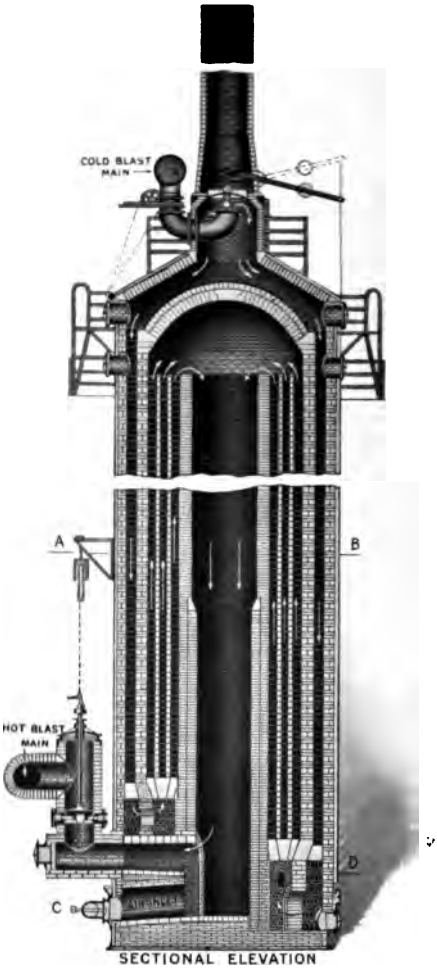
NOTE:
MADE OF 9" & 13½" BRICK

CALDER HOT BLAST STOVE
THE S. R. SMYTHE CO., Engineers, PITTSBURGH, PA.

FOOTE PATENT STOVE
D. LAMOND & SON, Engineers, PITTSBURGH, PA.

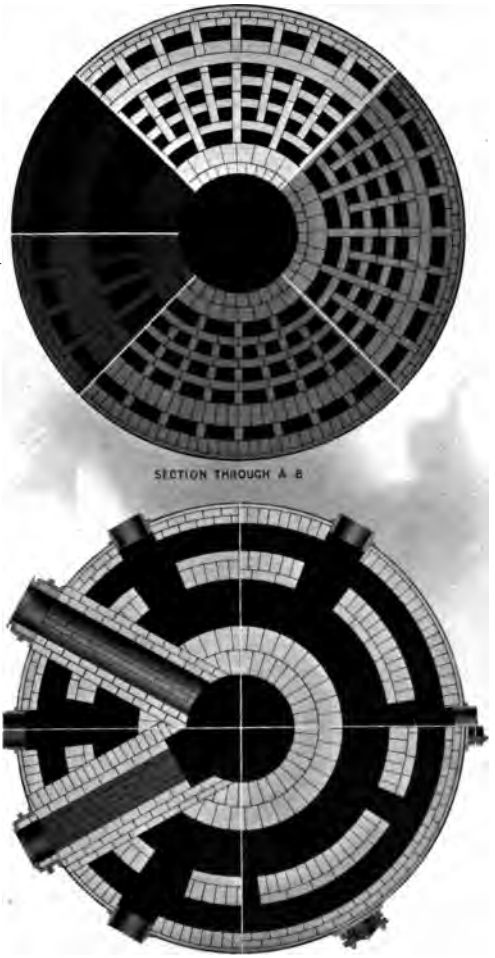


FOOTE PATENT STOVE
D. LAMOND & SON, Engineers, PITTSBURGH, PA.



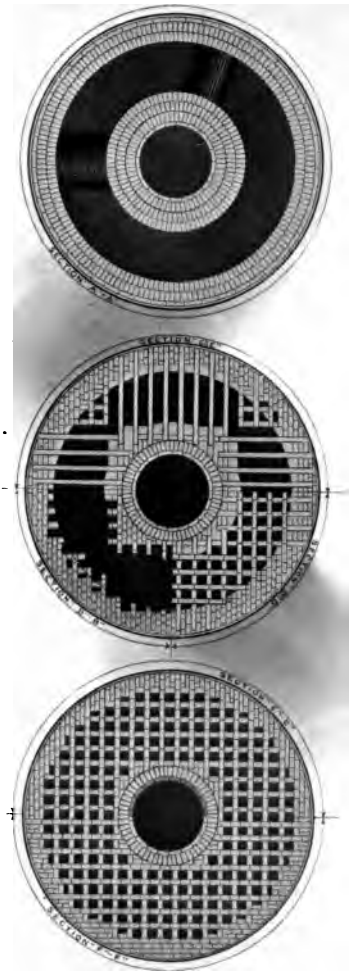
McCLURE STOVE

G. W. McCLURE, SON & CO., Engineers, PITTSBURGH, PA.

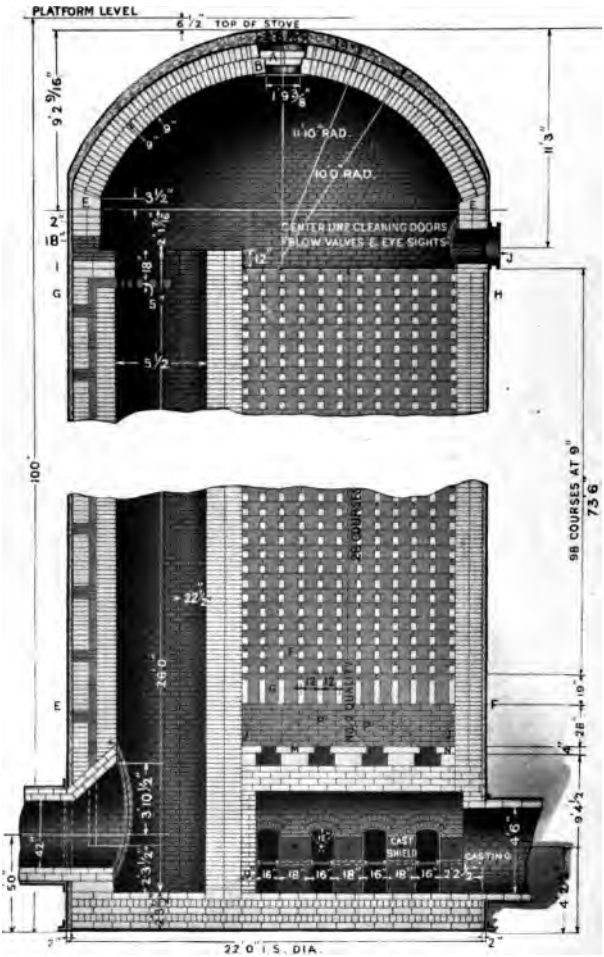


McCLURE STOVE
G. W. McCLURE, SON & CO., Engineers, PITTSBURGH, PA.

NELSON STOVE
ARTHUR G. MCKEE, Engineer, CLEVELAND, O.

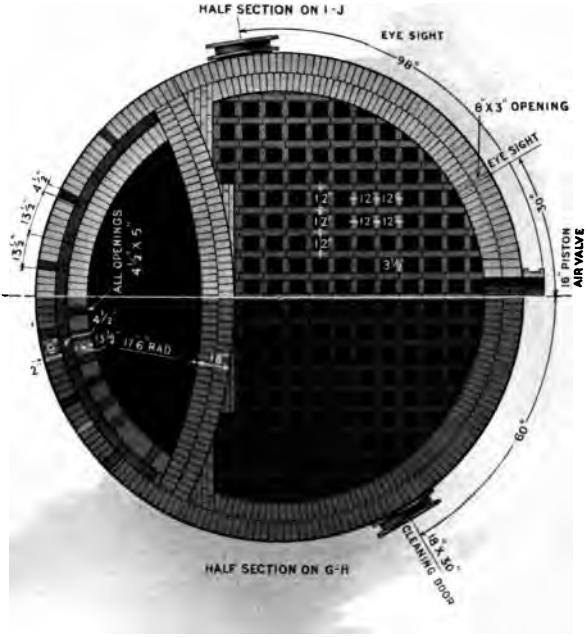


NELSON STOVE
ARTHUR G. MCKEE, Engineer, CLEVELAND, O.

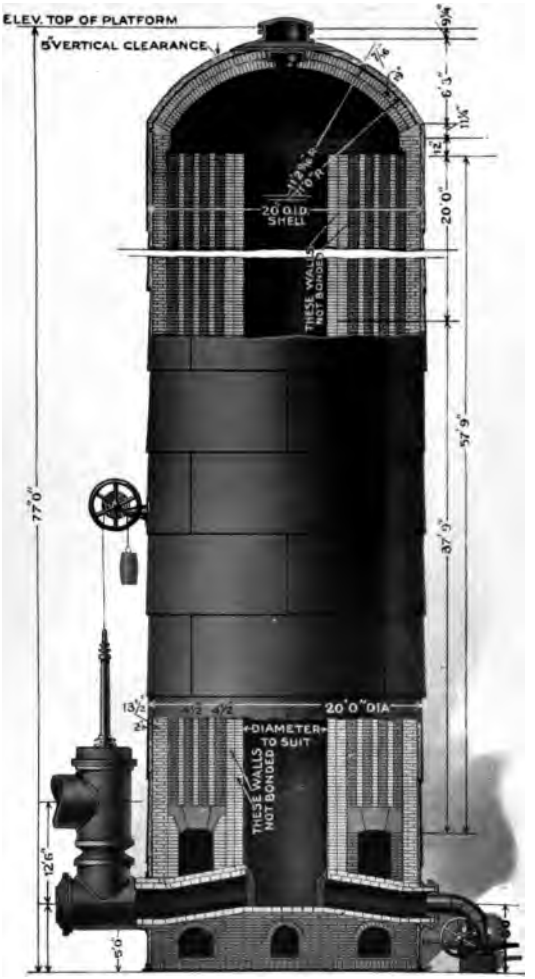


ROBERTS PATENT STOVE

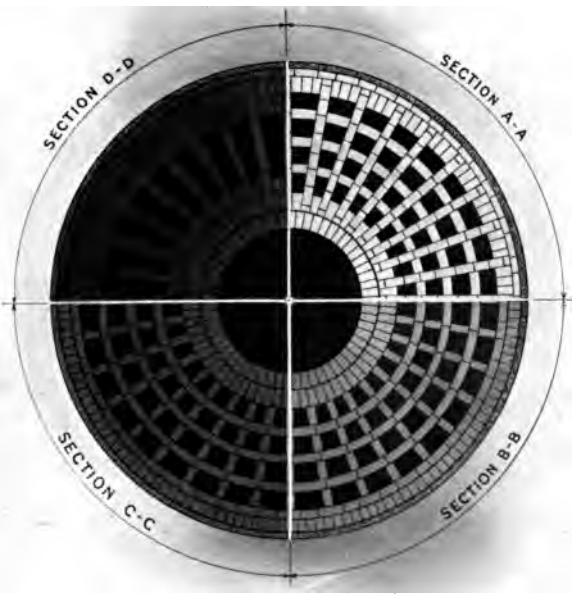
F. C. ROBERTS & CO., Engineers, PHILADELPHIA, PA.



ROBERTS PATENT STOVE
F. C. ROBERTS & Co., Engineers, PHILADELPHIA, PA.

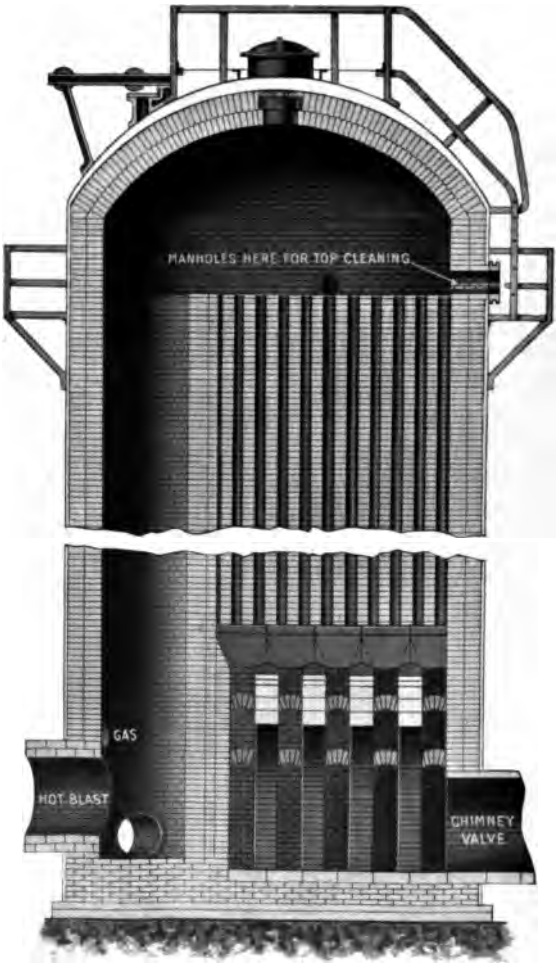


JULIAN KENNEDY STOVE
JULIAN KENNEDY, Engineer, PITTSBURGH, PA.

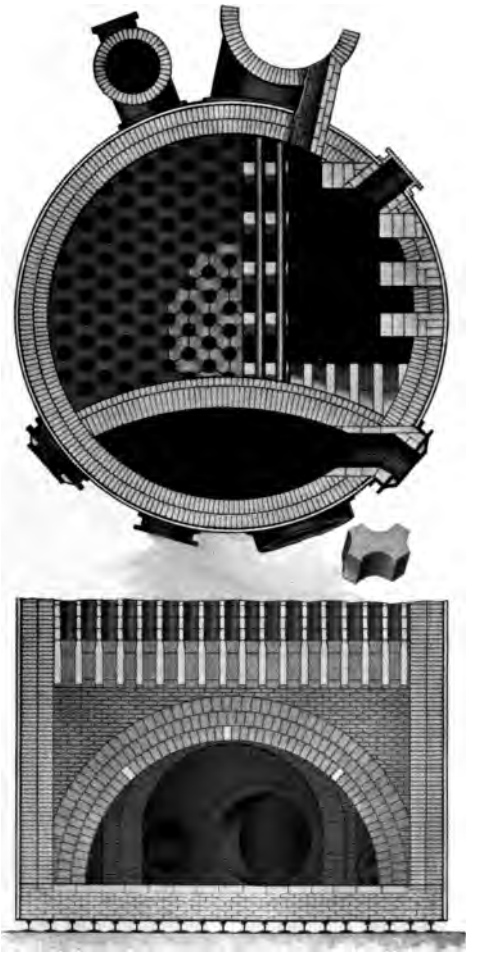


JULIAN KENNEDY STOVE

JULIAN KENNEDY, Engineer, PITTSBURGH, PA.



WHITE & KERNAN HOT BLAST STOVE
F. L. WHITE, PITTSBURGH, PA.



WHITE & KERNAN HOT BLAST STOVE.
F. L. WHITE, PITTSBURGH, PA.

the temperature of the outgoing gas from the stove and may be as high as 900° or 1000° F. although this is by no means usually the case, and shows that a large amount of heat is wasted, being unabsorbed by the checker brick. This valve may or may not be water-cooled and may be one of several types.

The hot blast valve must withstand often some 1500° F. and is also usually of the mushroom type, water-cooled, with its seat so arranged as to be easily renewable, and is lined throughout with a high grade of fire brick. It is subjected to by far greater wear than are the other valves.

In order that the blast may be taken off the furnace without stopping the engines, a valve termed the snort valve is usually introduced in the cold blast main and is operated from the cast house.

DISPOSAL EQUIPMENT

PIG BEDS — The previous description of equipment has been that for the production of the pig iron and its accompanying slag. The following equipment is that applicable to its disposition.

As the metal is tapped from the furnace, it is allowed to flow through a channel in the sandy bed of the cast house into the so-called "pig beds," or through runners into ladles. *It is apparent* that if the iron is to be used in *making steel* in a nearby open hearth or bessemer

plant, there is a marked saving in using the hot metal thus saving the re-melting costs. In such cases the molten iron is merely allowed to run off into a ladle car placed below the level of the cast house floor and then conveyed to the Bessemer converter or the open hearth furnace.

As the iron is tapped from the furnace, the first portion is usually comparatively free from slag, but later as the hearth is nearly emptied, slag mingles with it and it is necessary to separate the two. The old method was by molding a dam in the runner or trough leading from the tapping hole to the casting beds and between this dam and the furnace to suspend an iron plate or skimmer across the trough, the bottom of the skimmer being lower than the top of the dam. This effectually intercepts the slag, which is led off at one side. Present practice is to use a heavy cast iron trough of practically identical form with the molded sand dam. With the casting, much less skill is required and its use is considerably more satisfactory.

When the metal is to be cast into pigs in pig beds, the iron is directed along a depression or runner in the sand bed of the cast house, from which it flows into cross-runners, thence into the pig molds.

The pig molds are usually made in the *molding* sand of the floor with wooden patterns

and must be re-made after each cast. In some cases instead of being cast in the form of pigs, the number of cross-runners is greatly increased so that the iron is led into a series of long parallel "sows" as the metal in the cross-runner is often termed. These sows are then broken into the lengths of ordinary pig, either by hand or some form of machine breaker.

CHILLS — There is a disadvantage in such sand cast pig where it is to be used in the basic open hearth, due to the greater or less quantity of adherent sand which being highly silicious, makes with the necessary lime to flux it, a heavy slag. For this reason primarily the iron is sometimes cast in what are termed "chills." These are merely cast-iron pig molds, into which the metal is led, just as though casting in sand. Inasmuch as the only preparation of the molds is a washing with a clay grout or lime to prevent adherence of iron, a larger tonnage can be handled in a cast house of given floor space with chills than when sand cast.

CASTING MACHINE — With the increased size and output of modern furnaces, it is impossible to handle the enormous product as outlined above. This has led to the development of the pig casting machine where the casting is done from a ladle. Of the two types *of machine* the more modern and satisfactory *consists of* an endless chain carrying a series

of steel or cast iron molds parallel to each other with the edges overlapping. As the ladle is tipped, the travel of the chain brings into position a continuous train of empty molds to be filled. The chain carries the full molds through a trough of water, thus cooling the iron so that it may be dumped at the turn into cars, ready to be shipped, and on the return the mold is sprayed with lime or given other coating to prevent the metal sticking. There are a number of modifications to the above.

The other and older type of machine carries the series of molds on the periphery of a horizontal revolving wheel some 35′ to 40′ in diameter, but the upkeep of this type has apparently proven more expensive than that in which the endless chain is used.

SLAG DISPOSAL — Although a vast amount of research has been and is being conducted relative to economical and if possible, profitable uses for slag, by far the larger portion of it is as yet thrown away, the only problem being one of the cheapest disposition. It is usually allowed to flow into slag cars which are shaped somewhat like a thimble, often lined with fire brick, and the slag is then hauled to the dump and emptied while still molten. In many of the older plants there are yet in use small iron slag cars with a rectangular body in which the slag is allowed to cool, the blocks of slag being then dumped. Comparatively recently many

furnaces have installed equipment for the granulation of slag which, on being tapped from the furnace, is allowed to flow into a large masonry tank of water, or direct into steel cars with a jet of water playing under the slag as it falls from the runner. In this way it is broken up into particles similar to very coarse sand, and can then be handled from the tank to cars by some form of bucket and used as ballast or filling for which it is very satisfactory. This method also has the merit of cheapness. The manufacture of brick from blast furnace slag has been attempted in numberless instances, but these experiments have been successful in only a few instances with slag occurring in American practice. Failure is chiefly due to the slaking of the lime content on continued exposure to atmospheric conditions. The large and increasing use of certain blast furnace slags in the manufacture of Portland cement bids fair to make slag one of the most important by-products of the iron and steel industry. It also finds a somewhat limited use as a substitute for roofing gravel, etc.

MATERIALS—ORES

CHAPTER II

IT is difficult to frame a definition which shall be absolutely accurate from all points of view, and furthermore, usage has much to do with the various terms employed, but in general iron ore is a compound of iron combined with certain elements. This compound is more or less intimately mixed with earthy impurities or gangue as it is called. In order to be an ore in the sense here used, the iron must be in sufficient quantities and in combination with such elements as shall permit its smelting on a commercially profitable basis.

Although iron is one of the most widely distributed of the elements, it practically never occurs native or in the form of pure metallic iron, so great is its affinity for other elements. In general, the compounds of iron available for the blast furnace are the oxides, which include the three groups, the Hematites, the *Brown Ores*, and the Magnetites, these com-

prising more than 99% of the ore produced in this country, and the Carbonates, less than 1%.

The above are given in the order of their importance.

There is also the general division of all ores into two classes, i.e., Bessemer and non-Bessemer or basic, dependent upon the content of phosphorus. This terminology comes from the fact that in the acid Bessemer method of making steel, no phosphorus is removed during the process. Hence, the pig iron from which the steel is made must have no greater quantity of phosphorus than will maintain the Bessemer limit, .1% phosphorus in the steel. Practically all the phosphorus in the ore goes into the pig iron. This usually means that a Bessemer ore must contain no more than .045% to .05% phosphorus, the limit being .001% phosphorus for each per cent. of iron in the ore. Bessemer and non-Bessemer ores are in turn separated into other groups, depending upon their phosphorus content, but the two general divisions are as cited.

HEMATITES — These ores, often termed "Red Hematites," are anhydrous sesquioxides, Fe_2O_3, identical in composition with ordinary red iron rust, and if pure and without the gangue, would analyze 70% iron. They comprised during the year 1908 more than 88% of the total ore mined in this country. They give the characteristic red streak on porcelain.

The formation, as in the case of a majority of the iron ores, varies widely from a massive formation so hard that it can with difficulty be drilled and blasted, to the earthy, loamy formations which can be easily handled in open cut with the steam shovel. One of its forms is the familiar crystalline ore termed Specular Hematite, much used in open hearth practice; another, the Foliated form, often called Micaceous ore. In accordance with its various formations it may be locally known as Gray ore, Fossil ore, or Oolitic ore, etc. It is but very slightly attracted to the magnet. The range of composition in the Lake Superior district, by far the largest producer, is given for the year 1907, as follows:

	Per Cent.		Per Cent.
Fe	39.0	to	67.0
SiO_2	1.0	to	43.0
P	.01	to	1.0
S	.005	to	.14
Mn	.03	to	8.7

They occur both as Bessemer and non-Bessemer ores.

They occur as is shown on the map, (which is attached to the inner back cover), principally in northern Minnesota, the northern peninsula of Michigan, northern Wisconsin, Alabama and Tennessee. The so-called Lake Superior district in the northern part of Minnesota, Michigan

and Wisconsin supplies over 90% of the total Hematite ores produced.

There are here five great ranges, in order of their importance being Mesabi, Menominee, Marquette, Gogebic and Vermillion, the product of the last four being termed "old range" ores, to distinguish them from the Mesabi. As of interest, their relative tonnage for the last four years is shown:

	1907 Long Tons	1908 Long Tons
Mesabi	27,495,708	17,257,350
Menominee	4,964,728	2,679,156
Marquette	4,388,073	2,414,632
Gogebic	3,637,102	2,699,856
Vermillion	1,685,267	841,544
Miscellaneous	95,790	122,449
	42,266,668	26,014,987

	1909 Long Tons	1910 Long Tons
Mesabi	28,176,281	29,201,760
Menominee	4,875,385	4,237,738
Marquette	4,256,172	4,392,726
Gogebic	4,088,057	4,315,314
Vermillion	1,108,215	1,203,177
Miscellaneous	82,757	91,682
	42,586,869	43,442,397

The Mesabi ores are worthy of special note, both on account of their quantity and the effect *that their use* has had upon blast furnace *practice. This* is due to their formation, as

these ores are practically all of a loamy texture and of extreme fineness.

BROWN ORES — These are hydrous sesquioxides of iron and are distinguished from the true Hematites only by the presence of their water of combination. Approximately 7% of the total ore produced in 1908 belonged to this class. They are represented by the formula, $Fe_2O_3nH_2O$, and if pure would contain from 59.8% to 66% of iron, this amount depending on the degree of hydration. There are, as in the case of the Hematites, many varieties, Limonite being one of the most important ones, 2 Fe_2O_3 3 H_2O, and according to such formations, are locally known as Brown ores, Bog ores, Brown Hematites, Limonites, etc. If heated to sufficient temperature to expel the water of combination they change to true Hematites. In accordance with the degree of their hydration, the characteristic streak varies from yellow through brown to red.

Their formation is often massive but also occurs as concretionary, as veins in quartz or in pocket formation in residual clay deposits. Their occurrence is remarkably persistent throughout the entire eastern part of the Appalachian region, extending from New England to Alabama. A reference to the tables on pages 65, 66 and 67 will show Alabama, Virginia, Tennessee and Georgia to be by far the *heaviest* producers.

This formation as a whole is naturally separated into the two divisions of the Mountain and Valley Brown ores, and the Oriskany Brown ores of Alleghany County, Virginia, and Central Pennsylvania. The Mountain and Valley ores, although persistent, show no deposits of great size and are very irregularly scattered; the Oriskany ore, on the other hand, although of about the same grade, occurs in much larger deposits in well defined strata sometimes several miles in length and of very considerable thickness. This form of occurrence is important from the standpoint of production.

Due to the phosphorus present, practically all the Brown ores are of non-Bessemer grade, the average range of analysis being:

	Average Range in Composition of Mountain Ore.	Average Range in Composition of Valley Ore.	Average Range in Composition of Oriskany Brown Ore.
Fe	35.00–50.00	40.00–56.00	37.00–50.00
SiO_2	10.00–30.00	5.00–20.00	10.00–25.00
P	.10– 2.00	.05– .50	.06– .50
Mn	.50–10.00	.30– 2.00	.50– 4.00

MAGNETITES — The Magnetites, Fe_3O_4, comprise about 40% of the ore mined. It is the richest of the ores, having 72.4% iron when pure. It is very hard, heavy and dense, usually black or iron gray in color, of crystalline structure and possessed of marked magnetic *properties,* being nearly as magnetic as iron *itself. This* characteristic is all important if

its concentration is necessary, a condition later to be taken up. It often carries with it large percentages of Titanium, the Adirondack region being the heaviest producer of both non-Titaniferous and Titaniferous ores, the latter varying from a high grade, low phosphorous ore through the Bessemer grades into the non-Bessemer. Due to its crystalline formation and magnetic properties, many varieties lend themselves to concentration with comparative ease, and may so be changed from non-Bessemer as mined, to Bessemer, or low phosphorous ores, by reason of the removal of apatite, the phosphorous-bearing mineral.

Northern New Jersey is also a large producer, and southeastern Pennsylvania, near the vicinity of Cornwall, has most important mines, from which over twenty million tons have been taken. This latter formation is a peculiar one, as the Magnetite here occurs both in large and small masses, usually enclosed by sedimentary rocks instead of in igneous formations, as is generally the case. It is characterized by low phosphorus but is so high in sulphur, due to the presence of copper pyrites, as to require roasting. The presence of copper also produces a considerable percentage of this element in all pig iron made.

In Page County, Virginia, and in North Carolina occur Magnetites which are of Bessemer *quality.*

CARBONATES — The Carbonates, $FeCO_3$, yielding 48.2% iron, if pure, are least important economically, forming less than 1% of the ore mined. The varieties are variously known locally as Spathic ore, from the similarity in cleavage to feldspar, Kidney ore, Black Band ore, when associated with bituminous matter in coal veins, clay Iron-stone ore, from its frequent occurrence in beds of shale or clay, and as siderite, etc.

Due to their low content of iron and high percentage of carbon dioxide, which usually necessitates calcination before the ore is introduced into the furnace, Carbonates are of those least valuable. However, their association in occurrence with coal veins and the fact that their gangue is often self-fluxing, in a measure offsets the leanness of the ore. They are at present produced only in one state—Ohio.

In general, the Magnetites are geologically the oldest of the ores and of an entirely different formation from the rest, being metamorphic. The Hematites are supposed to be the results of alteration of the Limonites, the primitive ores, by the heat and pressure of the earth's crust.

The Carbonate ores are of still another formation, although on weathering, they may change to Limonites, on losing their CO_2.

As a general summary, the tables showing *the ore production* both by states and by varieties *will be of interest.*

PRODUCTION OF IRON ORE IN THE UNITED STATES BY STATES AND VARIETIES, IN LONG TONS.

1907

State	Hematite	Brown Ore	Magnetite	Carbonate	Total Quantity
Alabama	3,144,011	895,442	.....	.....	4,039,453
Arkansas and Texas	.....	118,667	.....	.....	118,667
Colorado	.....	11,714	.....	.....	11,714
Connecticut and Mass.	.....	37,166	.....	.....	37,166
Georgia	106,885	337,229	.....	.....	444,114
Kentucky, Md. and W. Va.	36,000	26,808	.....	.....	62,808
Michigan	11,830,342	.....	.....	.....	11,830,342
Minnesota	28,969,658	.....	.....	.....	28,969,658
Missouri	42,527	a 69,241	.....	.....	111,768
Montana, Nevada, N. Mexico, Utah and Wyoming	608,056	b 16,000	c 195,488	.....	819,544
New Jersey	.....	46,818	502,942	.....	549,760
New York	140,101	10,000	1,224,919	.....	1,375,020
North Carolina	.....	.....	50,439	.....	50,439
Ohio	.....	.....	.....	23,589	23,589
Pennsylvania	19,403	113,076	704,808	.....	837,287
Tennessee	269,182	544,508	.....	.....	813,690
Virginia	89,867	696,518	471	.....	786,856
Wisconsin	804,454	34,290	.....	.....	838,744
Total	46,060,486	2,957,477	2,679,067	23,589	51,720,619

a Includes Iowa b Includes Washington c Includes California

PRODUCTION OF IRON ORE IN THE UNITED STATES BY STATES AND VARIETIES, IN LONG TONS.

1908

State	Hematite	Brown Ore	Magnetite	Carbonate	Total Quantity
Alabama	2,775,903	958,535	.....	.....	3,734,438
Arkansas and Texas	.....	a 55,966	.....	.....	55,966
Colorado	.....	10,176	.....	.....	10,176
Connecticut and Mass.	.....	28,112	.....	.....	28,112
Georgia	1,248	319,812	.....	.....	321,060
Kentucky, Md. and W. Va.	7,491	45,744	.....	.....	53,235
Michigan	8,839,199	.....	.....	.....	8,839,199
Minnesota	18,652,220	.....	.....	.....	18,652,220
Missouri	77,400	21,014	.....	.....	98,414
Montana, Nevada, N. Mexico, Utah and Wyoming	354,012	b 68,565	95,872	.....	518,449
New Jersey	.....	.....	c 394,767	.....	394,767
New York	33,825	.....	663,648	.....	697,473
North Carolina	.....	.....	48,522	.....	48,522
Ohio	.....	.....	.....	26,585	26,585
Pennsylvania	26,162	73,001	343,998	.....	443,161
Tennessee	226,038	409,305	.....	.....	635,343
Virginia	64,323	626,910	990	.....	692,223
Wisconsin	730,743	3,250	.....	.....	733,993
Total	31,788,564	2,620,390	1,547,797	26,585	35,983,336

a Texas alone b Includes Washington c Brown Ore included in Magnetite

HARBISON-WALKER REFRACTORIES CO., PITTSBURGH, PA.

PRODUCTION OF IRON ORE IN THE UNITED STATES BY STATES AND VARIETIES, IN LONG TONS.

1909

State	Hematite	Brown Ore	Magnetite	Carbonate	Total Quantity
Alabama	3,176,416	1,144,836	.....	.....	4,321,252
Colorado	.....	10,000	.....	.....	10,000
Connecticut and Mass.	.....	38,272	.....	.....	38,272
Georgia	9,521	211,495	.....	.....	221,016
Kentucky, Md. and W. Va.	45,000	53,657	.....	.....	98,657
Michigan	11,900,384	.....	.....	.....	11,900,384
Minnesota	28,975,149	.....	.....	.....	28,975,149
Missouri	67,391	22,563	.....	.....	89,954
Montana, Nevada, New Mexico, Utah, Washington and Wyoming	542,744	143,834	79,838	.....	766,416
New Jersey	.....	(a)	a 543,720	.....	543,720
New York	53,187	.....	962,146	.....	1,015,333
North Carolina	.....	.....	61,150	.....	61,150
Ohio	.....	.....	.....	16,527	16,527
Pennsylvania	5,144	80,715	581,030	.....	666,889
Tennessee	298,818	358,977	.....	.....	657,795
Texas	.....	6,474	.....	.....	6,474
Virginia	72,955	762,937	1,955	.....	837,847
Wisconsin	1,061,931	5,505	.....	.....	1,067,436
Total	46,208,640	2,839,265	2,229,839	16,527	51,294,271

a Brown Ore is included in Magnetite.

The total imports and exports in long tons for the years noted are also shown indicating the bearing foreign ores have upon our production.

	1907	1908
Imports	1,229,168	776,898
Exports	278,208	309,099
	1909	1910
Imports	1,694,957	2,591,031
Exports	455,934	644,875

ITEMS AFFECTING VALUATION OF ORES — The value of an ore is dependent upon its richness in iron, its composition, i.e., the minerals associated with it, and its location, as well as its physical condition, and it is impossible to consider any one of these qualifications entirely apart from the others, so interdependent are they.

For instance, a very lean ore, one low in iron, may have considerable value due to the self-fluxing character of its gangue, and consequent low cost of smelting. On the other hand, a rich ore may have associated with the iron, elements so deleterious in effect upon the pig iron as to render it valueless without some method of removal of the impurities before smelting, which removal may be so costly as to render the ore unavailable.

In general, ores profitable to smelt will contain from as low as 30%, to as high as 60% *or 65% metallic* iron, in valuation each per *cent. being* termed a unit; thus, an ore with

50% metallic iron will have 50 units. It is evident that since iron is the thing of value, in any given ore it must be in such quantity as to bear not only the cost of the ore laid down at the furnace, but the cost of its extraction with profit. Furthermore, as the cost of reduction and melting any given quantity of iron as such, is constant, the variable enters in the disposition of the gangue.

From this it will be seen that the value of an ore increases not in proportion to the increase in the number of units of iron, but very much faster, as there is a corresponding decrease in the gangue to be handled.

As a whole, the gangue of iron ores consists principally of silica and alumina, together with small amounts of lime, magnesia and oxides of manganese and the alkalies, as well as more or less phosphorus and sulphur. So far as affecting the quality of the pig iron produced, the exact percentages of all the above, with the exception of sulphur, manganese and phosphorus are of little importance, as they are easily controlled by the burdening and operation of the furnace.

The sulphur, however, is but partly under control and requires for its even partial removal in the furnace, additional flux and fuel. Although no exact percentages can be given, as it is dependent on the analysis of the particular ore, the furnace operation, etc., any ore having *more than* .50% to .75% sulphur must be treated

to remove a portion before being charged. Such cost, together with the added expense of its smelting, will then determine whether or not the ore is available for even such pig iron specifications as allow comparatively high sulphur. It might still be used in a mix with low sulphur ore.

Manganese is another element but partially under control. It is not harmful to pig iron within the limits of, say 1% for Bessemer iron and 2½% for the Open Hearth. On the average, about three-fourths of the manganese in the ore will be reduced. However, if the manganese is sufficiently high to be, say, 18% to 20% or over in the pig, the ore then becomes suitable for the manufacture of speigeleisen, and with proper burdening, for the manufacture of standard ferro-manganese.

In phosphorus we have, more than in any other element, the controlling factor determining whether an ore is available from the standpoint of composition, and if so, what its classification. It must be assumed that practically all the phosphorus in the ore goes into the iron, as very little if any, can be eliminated in the blast furnace. This percentage of phosphorus leads to the two general divisions of Bessemer and non-Bessemer ores previously referred to.

LOCATION — The location of ore bodies is *naturally a most* important consideration. Their

location as regards fuel supply, means of transportation, and distance from the market, are among the determining factors. The size of the deposits will determine whether there is justification for the erection of a plant, although the wisdom of such an outlay will depend, in turn, wholly on the location of the present or future market for the output of the plant.

Mining costs, while low at the surface, are subject to advance as the work proceeds; in a shaft operation, the increased haul and difficulty of taking care of the water add their quota of expense; in a stripping proposition, the increasing over-burden as the ore strata dip means increased costs.

There also must be considered the natural exhaustion of the richer and more accessible deposits, tending each year to bring into effective competition ores of lower grades. All these items, as well as the character of the entire ore formation, are a few of the factors entering into the problem.

TREATMENT OF ORES

Although but a comparatively small proportion of the ores in this country require preliminary treatment before their introduction into the blast furnace, it is with certain classes advisable. The treatment usually comes under one of four heads: — roasting, calcining, concentration and nodulizing.

ROASTING — The object of roasting is the *removal of* sulphur and consists in heating the

ore in contact with air to as high a temperature as possible without incipient fusion. The sulphur usually exists in the form of iron pyrites, FeS_2, one atom of which is very easily expelled on heating either with or without contact with air, as expressed in the equation $FeS_2 = FeS + S$. To drive off the remaining sulphur, not only must there be sufficient heat, but an excess of air. This takes place in two stages:

$$FeS + 2\ O_2 = FeSO_4$$
$$2FeSO_4 = Fe_2O_3 + 2SO_2 + O.$$

In this way the sulphur may be reduced to but a trace of the original quantity present.

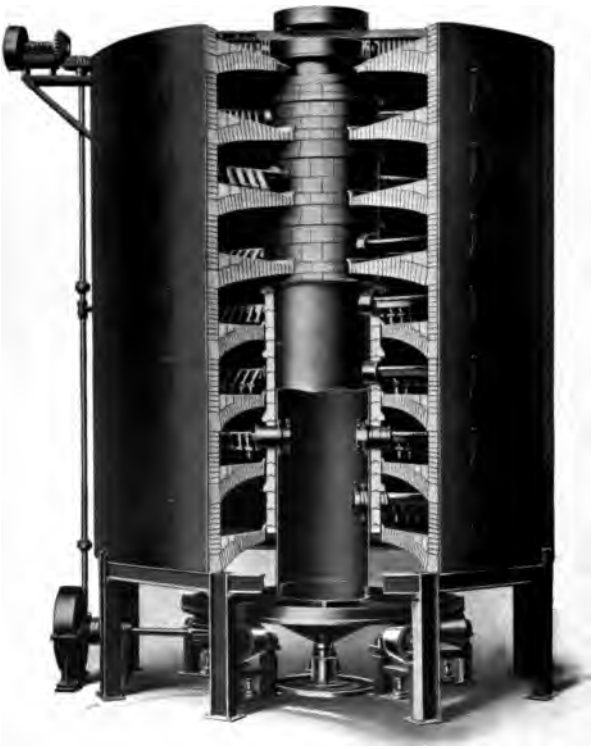
This operation may be carried on either in open heaps, in stalls or in kilns. The last is most economical and the temperature is under better control. Roasting in heaps is still practiced where fuel is cheap and there is ample space. A layer of coal a few inches in thickness is first spread on the ground, then a layer of ore, then coal, and so on, until a mass several feet in thickness is built up. The bottom is ignited and the whole heap slowly roasts. As an aid to the draft, the coarser ore is sometimes built into the bottom of the heap in channels. The heat may also be controlled in a great measure by the shape of the heap, whether deep and wide, or in comparatively long and narrow masses. Draft through the mass is controlled by the application of fine ore at any place where combustion seems to be too active.

A disadvantage is that certain portions of the heap may be insufficiently roasted while others where the draft is less impeded, due to coarser ore, etc., may easily get so hot as to fuse the surface of the ore and thus prevent thorough roasting.

Stalls, as their name indicates, are merely rectangular spaces enclosed on three sides by walls 8′ to 10′ high, through the bottom of which are holes for the draft. The ore is piled in these in much the same manner as in heap roasting. The defects of the latter method are present here, but not to so great an extent, as combustion is under somewhat better control.

In kiln roasting, a number of different types of kilns are used. One is the Gjers kiln, a vertical steel shell with a diameter of 15′ to 25′ and lined with 13½″ to 18″ of fire brick. The height varies with the diameter, ranging from 15′ to 35′, and the shell is somewhat contracted at the bottom, having holes for the admission of air and withdrawal of ore. The ore is mixed with coal and the kiln burns continuously, fresh charges being inserted at the top as the roasted ore is withdrawn from the bottom.

In a recent type of the Davis-Colby kiln, the ore descends in the form of a comparatively thin sheet, running the length of a rectangular kiln between two parallel flues of fire brick. These outer flues in which combustion occurs are fed by gas burners. The products of



WEDGE MECHANICAL FURNACE

WEDGE MECHANICAL FURNACE CO.,
GREENWICH POINT, PHILADELPHIA, PA.

combustion penetrate the descending body of ore through many openings and enter the inner flue, from this going to the stack. Either liquid or gaseous fuel may be used.

The illustration on page 74 shows another type of furnace — the Wedge Mechanical Furnace, which is built in various types, adapting it to the magnetic roasting and the desulphurizing of iron ores, as well as roasting of various ores and materials. The important points to be noted in the Wedge Furnace are uniform control of temperatures, protection of metal parts by brick (the 4′ diameter hollow central shaft being covered with tile attached to and revolving with the shaft), no loss of heat in making repairs, and low cost of operation and maintenance.

CALCINING — This is usually done with the object of making hard, dense ores, more permeable to the action of gases in the furnace, of removing the CO_2 from the carbonate ores, of rendering a non-magnetic ore magnetic as a preliminary to further concentration, or of expelling the water that may be present.

With certain ores, particularly the Magnetites, carrying silicates of iron in the gangue, this silicate is oxidized on calcination and the ore tends to become disintegrated. In many cases with other gangue material the ores are rendered more porous allowing freer access to the furnace *gases.* Continued application of a low red

heat to Magnetite, Fe_3O_4 also changes it to the form of Fe_2O_3, considerably more susceptible to reduction in the furnace, and calcination is employed in some instances with this object.

In the case of Carbonates, the equation on calcination with access of the air is:

$$2\ FeCO_3 + O = Fe_2O_3 + 2CO_2$$

In considering the calcination of the Hematites in order to render them magnetic, two methods may be employed. They may either be heated strongly in a neutral atmosphere with the result:

$$3\ Fe_2O_3 = 2\ Fe_3O_4 + O,$$

or,

$$3\ Fe_2O_3 = Fe_6O_7 + 2\ O.$$

Either of these oxides are strictly magnetic. The disadvantage of this method is that such high heats must be employed as to incur danger of fusion of the ore. They may also be heated at a comparatively moderate temperature in the presence of a reducing agent such as coal, or reducing gases occurring, for instance, in producer gas. In this method the ordinary type of vertical shaft kiln is usually available.

Brown ores, on application of sufficient heat, lose their water of combination and then become true Hematites and may be treated as such.

Carbonates, although changing to Fe_2O_3 on *heating, with* access of air, may yet be reduced

to the magnetic form by heating without access of air, the following equation occurring:

$$6\ FeCO_3 = Fe_6O_7 + 5\ CO_2 + CO$$

There is little occasion for such treatment, as Carbonates are of small importance as compared with the other ores.

CONCENTRATION — Concentration, as its name implies, always results in a richer ore, but its object may also be the removal of certain objectionable materials. Many of the ores which it is necessary to concentrate are of themselves far richer than others profitably smelted without concentration, but they are too high in certain elements or minerals, such as phosphorus, titanium, etc., which it is necessary to extract before they can be profitably handled.

Concentration usually takes the form of washing, jigging or magnetic concentration, the latter either wet or dry. Washing is applicable to the clayey ores and is conducted in one of the various types of washers in which the whole mass of ore is constantly stirred as it progresses through the washer, and running water carries off the clay, leaving the heavier ore.

In jigging, applicable to sandy or pebbly ores, the material is placed in boxes with perforated bottoms. These in turn are in tanks of water, and the constant jarring of the jig then agitates the material in the box and

allows it to settle in layers in accordance with its specific gravity; the ore goes to the bottom, while the pebbles and sand are allowed to waste as they accumulate and overflow.

Magnetic separation, whether wet or dry, depends upon the difference in magnetic permeability between certain minerals in the gangue of the ore and the magnetic oxide itself. It is, of course, immaterial whether the ore was originally magnetic or made so by subsequent treatment. The ore is crushed to suitable fineness and then subjected to the influence of magnets as it passes in a thin layer before them. This effects more or less separation. The magnetic portion of the stream is, in certain forms of separator, carried by a belt until it is out of the magnetic field, then allowed to drop into bins, while the portions which are unaffected by the magnets merely go to the tailings.

Separation by this method is essentially a grinding proposition, and it is in the grinding and screening that there is the larger portion of the costs, for the actual separation when once ground is comparatively inexpensive. The fact that such grinding separates the magnetic ore from the gangue is due to the crystalline formation, and the relative fineness to which any ore must be ground is in a great measure dependent on the structure of such ore, and the relative formation of the various

minerals as they are combined in this crystallization. The many details in connection with such separation, such as the suitable fineness of the ore and the type of separator adapted to it, are problems to be solved for each individual ore.

Wet separation is employed rather more abroad than in this country, and is a necessity with certain types of ore, where they must be ground to an almost impalpable powder. The principles involved are the same as in dry separation except that they are carried on with the ores mingled with water.

In general, this form of concentration finds a most useful application in the reduction of the phosphorus in Magnetites, and also in the reduction of the titanium. It bids fair to make available enormous deposits in New York which have hitherto been impossible to smelt profitably, due to the refractory nature of titanium-bearing oxides.

In typical ores the iron may be raised from 60% in the ore to 67% or even higher in the concentrates, and the phosphorus reduced from, say 1.75% to less than .70%, the tailings of apatite from such separation forming a valuable by-product. Such a statement, however, is of little value as the treatment of each ore forms a problem by itself, and one which can be *solved only* by actual tests.

NODULIZING -- The necessity for nodulizing is brought about by the difficulty of smelting very fine ores, either occurring naturally in such condition or made so by the processes of concentration. If very fine, the ore is easily blown about by the blast, and a very considerable percentage is carried away through the downcomers and distributed through the flues of the furnace plant.

The ideal ore should be uniformly lumpy, neither too fine nor coarse. Where both fine and lumpy ores are used, the tendency of the fines is to fill the voids of the coarser ore, producing a comparatively compact mass difficult for the gases to penetrate, and in a number of other ways causing furnace troubles.

This has led to a treatment of certain fine ores in nodulizing kilns which, in type, are nothing more or less than cement kilns.

The nodulizing kiln consists of an almost horizontal cylindrical steel shell approximately 100′ in length and possibly 8′ in diameter, lined with a high grade fire brick, this high grade material being especially necessary at the hot or discharge end. The kiln is placed at a slight angle with the horizontal and rotates during its operation so that the charge is continually working toward the lower end. At this discharge end a pipe is introduced conveying powdered coal which is ignited, giving intense heat, and the feed end of the kiln

is connected with a stack giving sufficient draft. The lining is usually 9″ to 12″ thick at the discharge end and from 4½″ to 9″ at the feed end.

The problem of the best brick for this work is one well worth the attention of the operator. It must, of course, be highly refractory and at the same time resist abrasive action. If too dense, it fails to properly take the coating of clinker and tends to spawl away in chunks as the kiln cools. If insufficiently refractory, it rapidly melts under the intense heats. It is also all important that the shapes used in the lining be well made in order to fit tightly against the shell, as the continuous rotation of the kiln will jar loose any lining improperly made or placed.

At the feed end of the kiln is introduced the ore, usually mixed with a slight amount of tar or pitch to agglomerate it. As the ore is fed and the kiln operated, the contents gradually work to a hotter and hotter zone and are finally discharged into conveyors, in the form of a sintered product varying in size from grains to small lumps or nodules. The size of these nodules is naturally determined by the details of the kiln operation. The process also is most beneficial as a desulphurizing agent. The material is now ready for the blast furnace.

At a number of plants, flue dust taken from the dust catcher and piping of the blast furnace

plant is thus treated with excellent results, as well as the iron bearing residues of copper ores from which the copper has been extracted.

BRIQUETTING — Briquetting is still another method employed for the same purpose as nodulizing. The fine ores are mixed with some adhesive as a binder, such as tar, rosin, molasses and various mineral preparations. The material is then pressed into some shape easily handled and thoroughly baked. In some instances, the ores resulting from wet magnetic separation are briquetted without other binder than their own moisture, but are subjected to such high temperatures in the drying kilns that this heat produces a sufficient bond. As is the case with nodulizing, briquetting also tends to desulphurize.

In general, the whole question of ore treatment, by whatever means, is most vital, and on its development depends in a great measure the practically unlimited iron production of this country. Due to advanced methods both of ore treatment and furnace operation, ores formerly valueless are today in demand. Vast quantities in the Superior region until recently considered too highly silicious to be of value, now bid fair to be large factors, due to processes of concentration recently perfected.

Although estimates of what may be the *quantity* of the ore bodies comprising the *available supply* must widely differ, due to the

varied assumptions made and the many indeterminate factors, the ultimate exhaustion of the known rich ore bodies within a comparatively definite period seems assured. Nevertheless, such progress has been and is being made in methods of concentration as to insure the availability of vast bodies of lean and impure ores, and places the date of their exhaustion far in the indefinite future.

MATERIALS-FLUXES & FUELS

CHAPTER III

FLUXES—Composition—As previously indicated, the flux is to render fluid the gangue of the ore and the ash of the fuel, a portion of the furnace charge which would otherwise be infusible at furnace temperatures, and is also to remove a portion of the sulphur in the form of calcium sulphide in the slag.

It is apparent then, that the nature of the flux is wholly dependent upon the nature of the material to be fluxed; the latter if an acid, requiring a basic flux, and vice versa. The vast majority of gangues are essentially acidic in their composition and consequently, require a base to form easily fusible compounds. In a few instances, the gangue itself contains such proportions of bases and acids as to render it practically self-fluxing.

The cheapest and most available flux is *limestone,* although in some instances lime in the form of oyster shells has been employed.

It is impossible to obtain a cheap and high grade limestone in all localities, consequently, recourse is had to one containing more or less magnesia or even to a true dolomite containing equal parts of calcium and magnesium. A charge part of limestone and part of dolomite is also often used.

With small quantities of magnesia, little difference is noted, but with larger amounts the question has often arisen as to whether it holds the sulphur in the slag as well as does the limestone. Although opinion is divided, the fact remains that in a number of furnaces and in widely separated districts the results with the use of dolomite are excellent both as to sulphur removed and fuel costs.

Limestone is a carbonate of lime, $CaCO_3$, together with small quantities of silica and whatever other impurities it may have. The oxide CaO being the active agent, CO_2 gas is really an impurity and this gas must be eliminated before the lime is in a position to perform its function.

Theoretically, this elimination can be most economically made outside the furnace by calcining, and the reason for repeated experiments along this line is evident when we consider just what occurs in the downward passage of the stone in the furnace when it is charged raw. As the stock descends subject to *increasing* temperatures, the stone attains *a dull red* heat at about 1100° F. and here

parts with its CO_2 gas, according to the equation:

$$CaCO_3 = CaO + CO_2$$

This gas rises with the ascending currents, but does not remain as CO_2, for at this heat the carbon in the coke has a marked affinity for the oxygen in the CO_2 and the following reaction takes place:

$$CO_2 + C = 2\ CO.$$

This CO passes off through the downcomers. In other words, for every pound of carbon in the limestone charged, an equal amount is abstracted from the furnace fuel and without performing any useful work.

Although calcination of the limestone would at first sight seem to overcome this, actual practice shows little if any advantage. This is due to the fact that CaO has a marked affinity for CO_2 at temperatures below a red heat, and if the flux is introduced into the furnace as CaO, it continues to absorb this gas in its descent, the CO_2 being present in the ascending volume of furnace gases. Hence, the CaO reverts to limestone, $CaCO_3$, and in turn, as it finally gets to a zone where the temperature is sufficiently high, the work of decomposition is repeated.

There is also a decided disadvantage in the use of the oxide due to its fine condition which renders it easily blown from the furnace when the blast is at high pressure, while raw lime-

stone crushed to pass from 4″ to 6″ rings is of ideal size for the furnace charge.

Experiments conducted abroad are said to show diminished fuel consumption and increased tonnage with the use of the oxide, but the actual economy when the cost of calcination is considered, is little, if any.

RELATIVE VALUE OF FLUXES — It is apparent that since the office of the limestone is to flux silica, alumina, etc., i.e., the acids in the gangue and ash, it also fluxes the acids present in its own composition, and its available base to apply in fluxing the gangue and ash is only whatever may be left after combining with the acids which it carries with it. From this is seen the importance of a stone low in impurities. Assume, for instance, limestone with the following analysis:

Acid		Base	
SiO_2	Al_2O_3	$CaCO_3$	$MgCO_3$
1.44	2.12	82.55	13.50

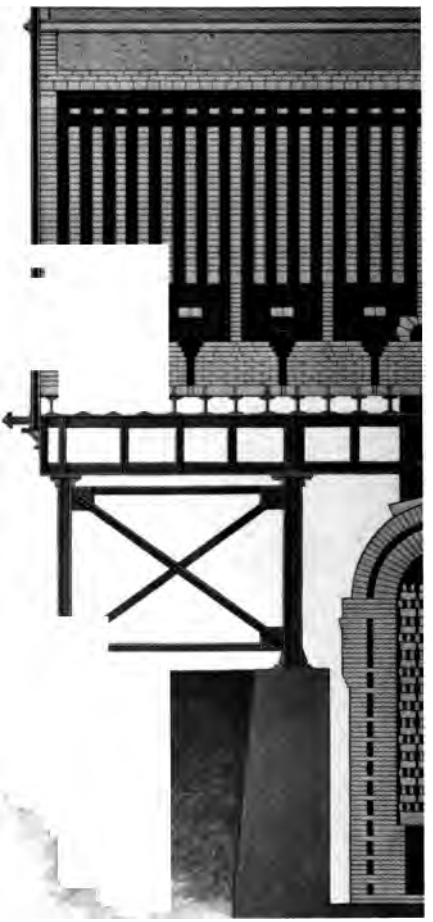
Of $CaCO_3$ the oxide CaO forms by weight 56%
Of $MgCO_3$ the oxide MgO forms by weight 47.6%

Therefore, $82.55 \times .56 = 46.23$ Total base in $CaCO_3$
$13.50 \times .476 = 6.43$ Total base in $MgCO_3$

52.66 Total base.
$1.44 + 2.12 = 3.56$ Total acids.

$52.66 - 3.56 = 49.10$

In other words, 49.10% is the available base of the flux and is the basis on which a *comparison* between various fluxes must be



BY-PRODUCT COI

UNITED COKE & GAS

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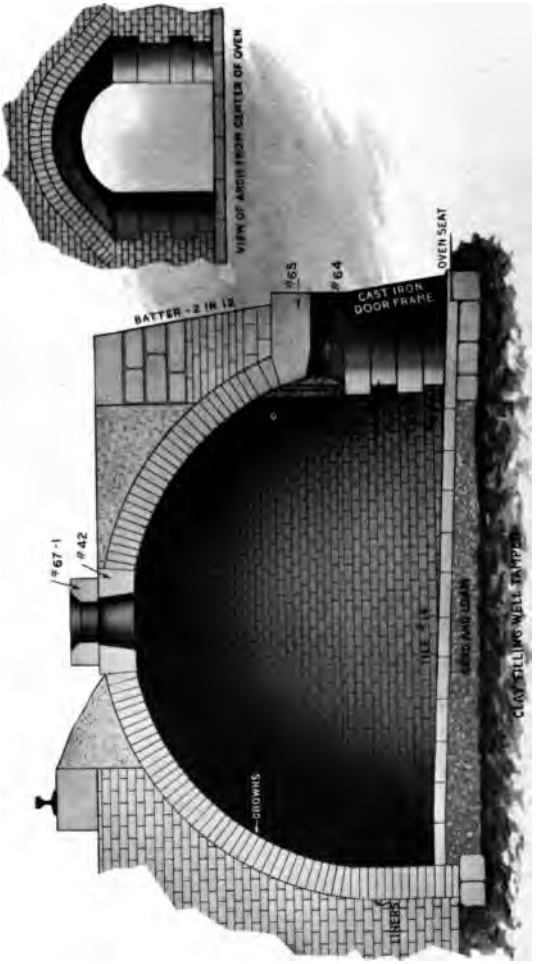
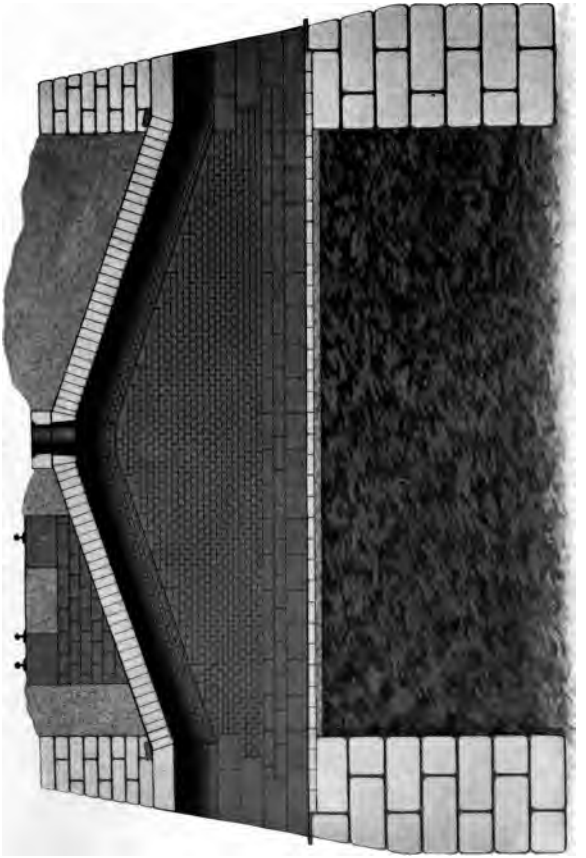
ITERY PLACE, NEW YORK



STANDARD BEEHIVE COKE OVEN



LONGITUDINAL SECTION

MITCHELL OR BELGIAN COKE OVEN

made, bearing in mind in comparing the quantity of CaO and MgO in any two fluxes that MgO is 1.4 times as efficient as CaO.

FUELS

There are practically but three fuels available for blast furnace work, i.e., coke, charcoal and anthracite coal. Of these, coke is by far the most important, fully 90% of the pig iron production in this country being made with that fuel, and the proportion is constantly increasing.

The prime essentials of a blast furnace fuel are:

1st. PURITY: — It must be high enough in fixed carbon to afford necessary fuel value, and sufficiently low in phosphorus and sulphur that the pig iron may not be unduly affected by these elements.

2d. PHYSICAL STRENGTH: — It must be strong enough to hold up its burden without crushing or softening at comparatively high temperatures.

3d. POROSITY: — It must be so open in structure as to allow sufficient area of combustion for the quick development of its heat.

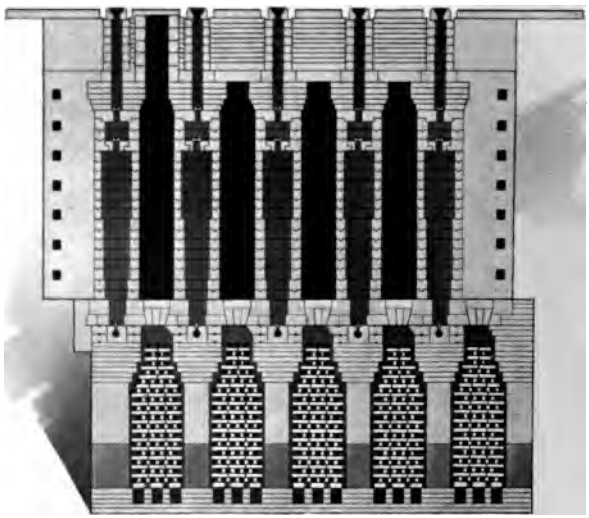
COKE — Coke is the residue resulting from the dry distillation of a coking coal. These coals are all bituminous but by no means all

the bituminous coals are coking coals, nor will anything but actual trial prove conclusively whether or not a coal has such properties. The analyses of two coals may be practically identical, yet one refuse to coke while the other is entirely satisfactory. A mixture of a coking and non-coking coal may also give good results.

Coking consists essentially in driving off the volatile constituents of the coal, mostly hydrocarbons, and a portion of the sulphur, the coke residue being the fixed carbon and the ash, with whatever sulphur and phosphorus may remain. In coking, the coal softens, swells, and becomes a viscous mass through which the bubbles of gas find their way to the surface and escape, leaving a hard, porous, open structure.

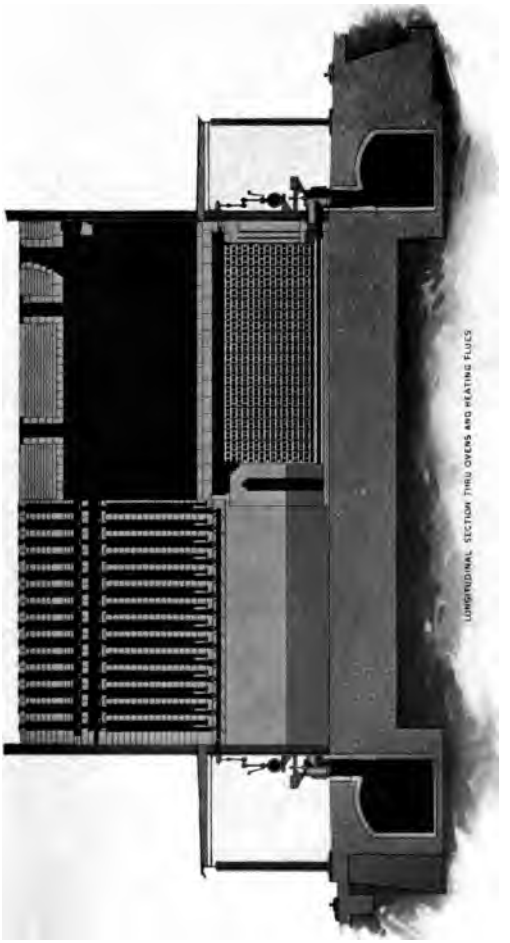
Inasmuch as the volatile matter in some coals will run as high as 37% to 38%, (although 18% volatile coal has been coked) it is apparent that there is a great shrinkage on coking, a ton of coal making in the Beehive oven approximately six-tenths to seven-tenths of a ton of coke, with the impurities of one ton of coal concentrated in the lesser amount of coke. This is not strictly true as regards the sulphur, as a portion of this is driven off during the process.

Impurities — The greater portion of the sulphur is usually in the form of iron pyrites, FeS_2, although some may be as a sulphate, in

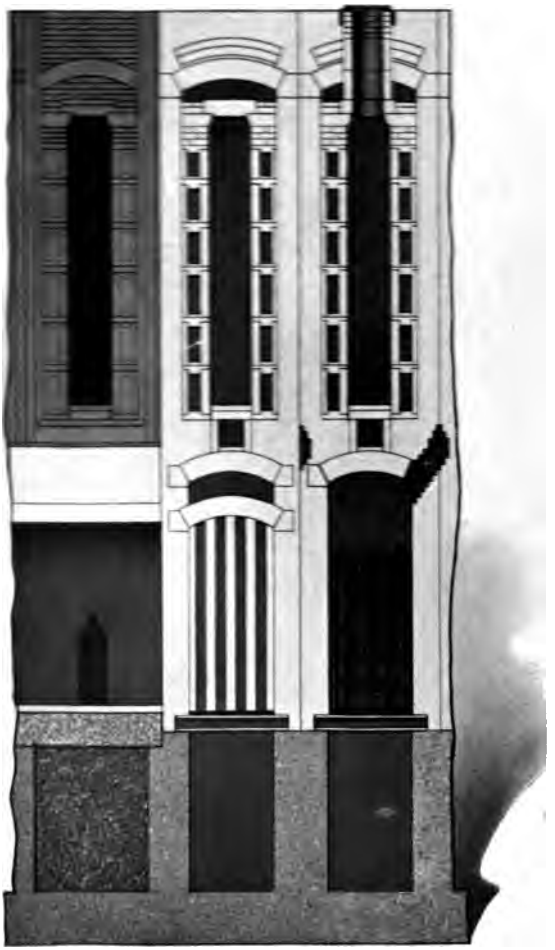


CROSS SECTION THRU. OVENS

KOPPERS BY-PRODUCT COKE OVEN

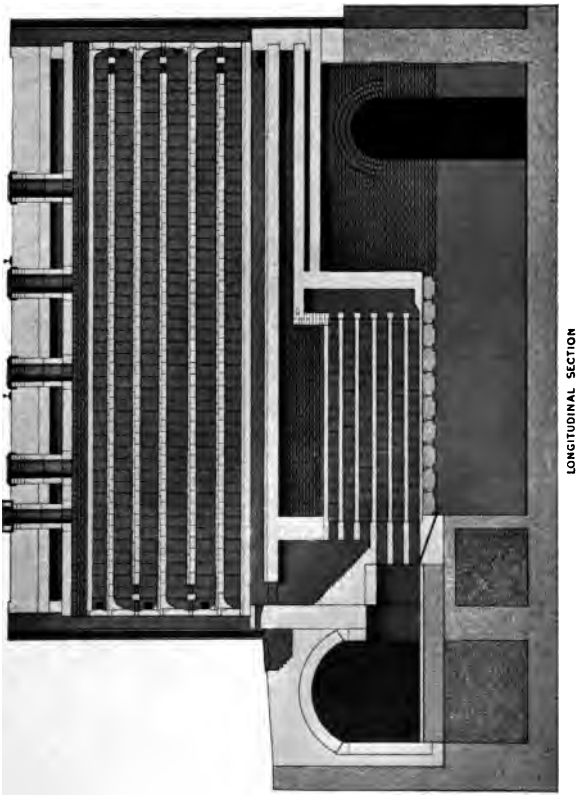


KOPPERS BY-PRODUCT COKE OVEN



VERTICAL CROSS SECTION

SEMET-SOLVAY BY-PRODUCT COKE OVEN



SEMET-SOLVAY BY-PRODUCT COKE OVEN

which form it remains unchanged in the ash. In the Pittsburgh district it is usual to consider about 90% of whatever sulphur there may be in the coal, to be in the form of iron sulphide or pyrites, and of this theoretically, five-sevenths may be eliminated in the coking process. From this a close estimate of the resultant sulphur in the coke may be made.

Many coals today too high in sulphur and too bony, i.e., mingled with slate, can be ground and washed, giving a much purer coal and also distributing the ash so that the physical strength of the coke is improved.

Phosphorus exists usually in the form of phosphate of lime and remains unchanged, obviously being of higher percentage in the coke than in the coal.

Coke is made in three types of ovens, i.e., the Beehive, the Mitchell or Belgian, and the By-product oven.

MANUFACTURE — *Beehive Oven* — As its name implies, this oven resembles a beehive, being nothing more than the section of a vertical cylinder about 12'6" in diameter, and say 2'8" in height, surmounted by a section of vertical dome as indicated on page 90. The total height of the interior is from 6'6" to 7'6". The oven walls are preferably made of high grade fire clay brick, made so as to withstand not only the heats encountered but the exceptionally *severe abrasion* especially incident to the *machine-drawn* oven. The crown of the oven

should be of a lime bond silica brick, exceptionally refractory and one which the experience of years has proven to be by far superior to any other material.

At the front of the oven is an opening or door to allow the withdrawal of the coke.

These ovens are built side by side in long batteries or banks, as they are sometimes called. A suitable loam filling then entirely covers the oven and dome, so that the whole battery forms one continuous block. Alternate ovens are charged and drawn so that each oven when freshly charged is fired by the heat of adjoining ovens already in blast.

The door is partially bricked up and the oven is charged with coal from a larry through an opening in the crown, the amount of such charge depending upon whether the coke is what is called 48-hour, 72-hour or 96-hour coke, i.e., in accordance with the period of burning. The longer the time of coking, the better the product in every way, the volatile matter is more thoroughly eliminated and the physical structure strengthened. The majority of Beehive blast furnace coke is a 48-hour product.

The charge is then leveled and the door bricked to the top with the exception of a small space to afford draft, or as it is termed "air" the oven. The outside of the door is then luted with clay to render it air-tight. The heat stored in the oven walls and that transmitted from adjacent ovens soon raises the temperature sufficiently to ignite the gases,

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which begin to distill. This still further raises the temperature and the whole mass becomes pasty. There is at first a marked swelling with the fusion, but finally, as distillation progresses, the mass shrinks to a volume considerably less than the original. As the process is finally completed, the door is torn down and the oven watered, thus causing the bed of coke to contract and split so that it can be withdrawn.

The structure of the coke is distinctly columnar, being 18″ to 20″ long in the direction of the depth of the coke bed. Good Beehive coke has a pronounced silvery white luster and throughout its mass are shiny nodules formed during distillation by the deposition of the carbon from hydrocarbons. There are also often fine jet-black hairlike filaments from the same source, an indication of good coke. The essential points to watch are the sulphur and phosphorus content and the physical structure of the coke.

The following are fair average analyses of dry Beehive coke in the districts indicated. These analyses are the average of a week's run.

District	Vol. Per Cent	Fixed C.	Ash	S.	P.
Connellsville..	.58	89.53	9.89	.833	.012
Latrobe.....	.30	88.05	11.65	.949	.031
Uniontown...	.38	89.87	9.75	.643	.013
Klondyke.....	.48	88.92	10.60	.848	.009

At a number of plants the gases, instead of being taken off as previously indicated, at the trunnel, are conducted through an opening at the back of the oven just above the spring line of the crown and are led through waste heat flues, to the boilers where they are burned. This service is particularly severe on the brick for lining but the employment of a high grade lime bond silica has in every instance given excellent results and seems to have entirely solved this problem. With this difficulty out of the way, there is no question but that the practice will largely increase, due to its very considerable economy.

A fair average of the gases from a Beehive oven on a Connellsville coal will show:

CO_2 — 3.0%
CO — 9.0%
H_2 — 11.0%
CH_4 — .30%
N_2 — 76.7%

The analysis varies widely with the airing of the oven, but the above may be assumed a fair average in good practice. In quantity, the amount is theoretically sufficient to supply approximately 27 H.P. per oven, and actual results have given over 19 H.P. per oven.

Mitchell, or Belgian Oven — The Mitchell oven is a modification of the Belgian type, one difference being that it provides an improved *combustion* chamber. The process is essentially

identical with that of the Beehive oven, but structural differences in its shape are evident as on page 91. In this type there is an opening at both ends; consequently, the oven can be discharged in a few minutes by a pusher extending through the oven, thus utilizing practically all the heat for the next charge. The oven is immediately charged after pushing.

Although comparatively new in the United States, this type bids fair to show very marked economies in operation and with greatly decreased loss in braize, or powdered coke, in which there is necessarily considerable waste in drawing any type of Beehive oven.

By-product Oven — In this type of oven, as the name implies, special attention is given to the utilization of the by-products in the gases and the gases themselves, after the by-products are extracted, are available for fuel. There are a number of types working successfully, such as the Koppers, the Otto Hoffmann, the Semet-Solvay and others. The illustrations on pages 88, 89, 94, 95, 96 and 97 will show clearly the outlines.

In general, the block of ovens form a series of narrow coking chambers or retorts from 16″ to 24″ in width, 5′ to 6′ high and possibly 30′ long, with a door at each end. The walls of the coking chamber form flues through which a portion of the gases of distillation pass and on *being burned* heat the coal as it is freshly *charged.*

In the By-product oven, the coal being in a comparatively thin body, is more quickly coked than in the Beehive oven and can be pushed as in the Belgian oven, thus allowing little heat to waste.

Since the coal cokes from the flue walls toward the centre of the mass, the gases escape here, forming a cleavage plane so that the coke is a little less than one-half the length of the width of the coking chamber. It is of a dark grey color and usually lacks the luster of Beehive coke. Its analysis usually shows somewhat higher fixed carbon than Beehive coke and it is claimed to be physically as strong, if not stronger. There has undoubtedly been some prejudice against it in this country, but a large number of furnaces are now operating very successfully with this form of fuel and some of the most important plants are using it exclusively. If properly made, By-product coke is probably more successful than Beehive coke in modern blast furnace practice, due to its greater regularity in physical structure and chemical analysis.

The gases are drawn from the top of the ovens and treated for the ammonia, tar, etc., they contain, all of which is wasted in the Beehive and Belgian ovens. Where the original costs of installation can be borne and where a plant is situated favorably as to market, there *is no* question whatever of the enormous

economy of the By-product oven over the Beehive. There is little doubt that its use is extending and that it will ultimately displace the Beehive oven in this country as it did abroad many years ago. Had it not been for the seemingly inexhaustible deposits of the highest grade coking coals, such as those of the Connellsville fields, and the large returns from the comparatively crude plants erected, operators would doubtless have more quickly realized the benefits of recovering the by-products of this as well as other industrial processes.

CHARCOAL — This fuel is the result of the dry distillation of wood, i.e., its heating without access of air. It was formerly made by placing the wood in heaps and covering with earth, leaving only sufficient openings to secure a slight draft. In this way the heat of combustion of a small portion of the stack serves to distill the volatile matter from the remainder, and the residue of almost pure carbon remains. In certain districts this process is still followed.

Distillation is now usually performed in closed retorts, securing a higher yield of charcoal, and the volatile portion is treated for the recovery of wood alcohol, tar, etc.

The charcoal preserves the shape and structure of the original wood but loses greatly in weight. It is extremely light, of exceptional porosity and purity, being low in ash and phos-

phorus and with no sulphur. An average analysis of charcoal is given as follows:

Fixed Carbon	Volatile Matter	Moisture	Ash	P.
87.25%	6.08%	5.77%	0.92%	0.016%

It is very friable as compared with coke and can support the burden of only a comparatively small furnace stack.

Charcoal was at one time used throughout the country when the adjacent timber land could furnish fuel for each local furnace, but with the advent of large operations and with the added cost as the timber disappeared, its use has very rapidly declined until the tonnage made on this fuel is a small one. On account of its exceptional purity it is still employed in the preparation of special charcoal irons in a number of furnaces in the southern and northern central portions of this country, where timber is yet available.

ANTHRACITE — This coal is used in its natural state and has some excellent characteristics as a blast furnace fuel. If properly selected, it is high in fixed carbon, low in ash, sulphur and phosphorus and so strong as to take any burden. The analysis varies somewhat widely with various districts from which the coal may be taken, but an average of good selected anthracite will show about as follows:

Fixed Carbon	Volatile Matter	Moisture	Ash	Sulphur
82.66%	3.95%	3.04%	9.88%	.46%

Its great defects, however, are its lack of porosity and its tendency to decrepitate or break up in small pieces as it is heated in the furnace. The first leads to the comparatively slow development of its heat, and the second to interference with proper passage of the gases in the furnace. Almost no stacks are using it entirely, although a few mix anthracite coal with coke. Even these are slowly being forced to the use of coke. It is obvious that on account of its high cost, anthracite coal is available only where there is a comparatively short haul, and its consumption is confined strictly to its own field of production.

VALUATION OF FUEL — Since the office of the fuel is primarily to furnish heat, it would seem at first sight that such value would be directly proportionate to the fixed carbon. This would be true, were it not for the fact that the ash and sulphur of the fuel must be removed by a flux, and the melting of this ash and flux requires heat. The *available* carbon, then, is that portion of the fixed carbon remaining after deducting whatever carbon is necessary to melt the slag formed by the ash and sulphur, and the flux required to do this. It is commonly estimated that for every pound of ash there will be required two pounds of stone, and for the sulphur, stone equal to three and one-

half times its weight. Assume, for instance, a coke analyzing:

Fixed Carbon.....86.50%
Ash.....11.50%
Sulphur..... .75%

11.50 × 2 = 23% stone for ash.
.75 × 3.5 = 2.62% stone for sulphur
25.62% stone

If the stone shows, for instance, 95% $CaCO_3$, 44% of 95% = 41.8% is CO_2 — a gas; consequently, 58.2%, the balance of the stone, enters the slag. In other words, in the stone to flux ash and sulphur there will be:

25.62 × .582 = 14.91%

Hence, total weight slag:

11.50% ash
.75% sulphur
14.91% stone
27.16%

or 27.16 pounds per 100 pounds of coke.

Experience shows that to melt the slag requires about 25% of its own weight in carbon.

25% of 27.16 = 6.79

Available C = Fixed C (86.50)—6.79 = 79.71%.

In this manner, the relative value of any fuel is easily determined.

BURDENING THE FURNACE

CHAPTER IV

INTRODUCTORY

THE burdening of a furnace is the determination of the proper proportions of ore, flux and fuel for the furnace charge and is directly dependent,— first, UPON THE ANALYSIS OF THE MATERIALS AVAILABLE TO CHARGE INTO THE FURNACE, and second, UPON THE CLASS OF IRON TO BE PRODUCED, whether basic, bessemer, foundry, malleable, low-phosphorus, forge, high silicon or whatever its character may be. The ability to burden properly was deemed one of the "gifts" of the old-time founder or operator, regarded with awe by the uninitiated, and oftentimes the results accomplished, in view of the inexact information at hand, were little short of wonderful. The modern laboratory, however, with its quick and accurate analyses, has done away with much of the old-time guess work, but nothing will do away with the necessity for judgment, experience and a certain

intuition which yet seems the possession of the most successful furnacemen. Theory and the laboratory have their places in the operation of the blast furnace and most important ones they are, but they are by no means all-sufficient. Success can be attained only by trial and application of intelligence and experience to the ever changing conditions of the furnace. The furnace must work freely and regularly and must produce the kind of iron desired. These results are accomplished by proper temperature and by proper composition of the slag regulated by the burdening of the furnace, and the two are by no means independent of each other.

SLAG — Since everything charged into the furnace appears either in the iron, slag, or gases, it is evident that all the non-volatile constituents in the ore, stone and coke which are not to appear in the iron must appear in the slag. The problem then, is to form such a slag as shall have for any impurities which might enter the iron, a greater attraction than has the iron, and by a proper composition of slags and regulation of temperature these results may be widely varied.

Blast furnace slags are essentially silicates of lime, the following cited from Campbell being typical examples:

Composition of Blast Furnace Slags

Slag							Iron		
SiO_2	Al_2O_3	CaO	MgO	FeO	S	Total not includ. S.	Si	S	Remarks
Averages for hot furnaces:									
33.21	13.67	40.68	11.08	....	....	98.64	3.79	tr.	Cuban Ore
34.84	11.75	41.30	9.79	....	....	97.68	2.46	.025	Spanish Ore
31.77	11.98	45.58	9.05	....	....	98.38	1.27	.020	Spanish Ore
35.55	12.05	40.52	8.86	0.68	1.66	97.66	1.79	.027	Lake Ore
Averages for moderate or cool furnaces:									
33.15	10.27	45.57	9.81	....	....	98.80	0.88	.07	Cuban Ore
30.73	11.32	47.36	8.35	....	....	97.75	0.35	.03	Spanish Ore
34.73	11.30	40.12	10.86	1.26	1.40	98.29	0.81	.063	Lake Ore
35.35	14.43	29.69	20.71	....	....	100.18	1.61	.040	Lake Ore

These analyses do not include, however, a number of extremes which have nevertheless given good results in continuous operation.

From these it will be seen that there is a very considerable latitude in the variation of the constituents.

Although the slag is, as mentioned, a silicate of lime, either the lime or silica may be and usually is, partially replaced by other radicals. Few limestones used as fluxes are so pure as to be free from magnesia and on the other hand, many furnaces use magnesian limestone, and some, a true dolomite partially or even entirely as a flux. Consequently, the lime of the slag is usually partly replaced by magnesia and very small quantities of iron, manganese and the alkalies. The silica is also always accompanied by alumina in greater or less proportions and by more or less sulphur acting in the form of calcium sulphide, CaS, as an acid radical. Inasmuch as none of the elements can enter the iron except in the reduced condition, we can determine at the outset which will be present by the test as to whether or not they are reduced or deoxidized under blast furnace conditions. As the lime, alumina, magnesia and whatever alkaline bases may be present are not reducible, whatever amount is in the ore, stone and coke will be in the slag; the same is true of the major portion of the silica, although some of it is reduced to silicon and enters the iron. Just how

much is dependent on the operation of the furnace and the character of the slag. Sulphur also combines with the iron to a greater or less degree, in accordance with the temperature of the furnace and the character of the slag. The same is true of manganese. Practically all the phosphorus in the charge will be found in the iron. There are then, only the silicon, sulphur and manganese, over which there is any regularity in the control.

COMPOSITION — In considering the constituents indicated in the previous analyses of slags, there is little difficulty in placing them definitely as acids or bases, with the exception of alumina. It occurs in slags apparently sometimes on one side of the dividing line, sometimes on the other, but in its effect, is so nearly neutral that any error, if error there be in assuming it to act as an acid with the silica as is usually done, is of comparatively small moment.

The sum of the silica and alumina is usually taken as the measure of the slag's acidity and with this assumption a reference to the analysis shows most slags as basic compounds, although this is by no means always the case.

As a general rule, the sum of the silica and alumina will range somewhat under 50% of the slag, alumina running usually about one-third of this amount, the exact ratio of the one to the other being dependent on the original

amounts in the flux used; but the proportion of the slag as a whole which they make up is dependent on the amount of stone used in the charge. With the sum of the silica and alumina constant, there may be a very considerable variation in the relative proportions of the two without marked changes in the characteristics of the slag, although as a whole a basic slag exhibits more marked basicity with high alumina and low silica than with low alumina and high silica. The converse is also true.

The sum of the basic constituents, lime and magnesia, usually forms about one-half the slag. Here again the ratio of one to the other is dependent on their ratio in the flux except as somewhat affected by the lime and magnesia content in the gangue of the ore and ash of the coke.

As in the case of alumina combined with silica, so there may be here considerable variation in the ratio of magnesia to that of lime without marked change in its characteristics, provided the sum of the two remains about the same. However, with high alumina and high magnesia both occurring, the slag usually tends to become too viscous on account of its extreme refractoriness.

CONTROL — The slag and the temperature of the furnace have previously been referred to as the agents for the control of silicon, sulphur and manganese, and the inter-relation of the

two may again be emphasized. In general, the more basic a slag, the more refractory it is, although it is also true that a highly acid slag is generally refractory. There is, then, a means for decreasing the fusibility of the slag by increasing its basicity, i.e., increasing the proportion of flux. At first sight it may seem that if the hearth is too cold, the temperature might be raised without regard to the composition of the slag, but this is true only within comparatively narrow limits, for the iron, acquiring most of its heat in passing through the hot slag, is in a great measure dependent upon it for any increase in temperature. An easily fusible slag is naturally but a short time within zone of fusion, quickly melting and forming a layer of slag over such iron as may be in the hearth and effectually prevents the transmission of any large quantities of heat through it, however hot the furnace may be at or above the tuyeres. With a more infusible slag, on the other hand, and one consequently longer in the heat zone, it absorbs more heat in its passage to the hearth and the iron trickling through it in a molten condition, the temperature of the iron is correspondingly raised. The relation of temperature and slag composition is an inseparable one.

Both the above, i.e., increased basicity and high temperature, have a marked influence on the control of the silicon and sulphur, a high

temperature in general favoring the reduction or deoxidation of silicon and its incorporation in the iron. However, the basic slag has a strong affinity for the silica and tends to neutralize the effect of temperature, keeping the silica unreduced in the slag. The basicity of the slag also has the effect of holding a large portion of the sulphur in it, as in the case of an open hearth slag, and results in low sulphur in the pig iron. Hence the final result will be, in general, low silicon and low sulphur in the pig with a high temperature and decidedly basic slag, and a constant increase in both silicon and sulphur as the slag changes in its scale from basic toward acid properties.

The combining limit of sulphur in basic slags is usually considered to be not over 2%, although some instances can be cited in which the slag averaged during considerable periods 2.4%, often running as high as 2.8% in sulphur. Because of the limit of solubility, it is essential that the slag be sufficient in volume. This is the case in the gangue of a large majority of ores, but where the slag is scant, a silicious ore, mill cinder, or even sand may be used, and its acidity neutralized by additional limestone.

The control of manganese is affected by the temperature and composition of slag, as is silicon and sulphur. A high temperature favors its reduction and, being a definite base, a highly

basic slag has less hold upon it than has an acid slag and allows more of the metal to be reduced and enter the iron, while with acid slag it is obvious there is less tendency for the manganese to appear in the pig. It is of marked benefit in the removal of sulphur, forming a sulphide MnS, thus decreasing the sulphur in the iron. This property is utilized, as has previously been shown, in the removal of sulphur in open hearth slags.

DETAIL OF PROBLEM IN BURDENING — In order to work out the theoretical burden of the furnace, it is of course necessary to have accurate AVERAGE analyses of the component parts and an idea of the analysis of the iron it is desired to make.

For the sake of illustration it is assumed that the pig iron desired will be, say 94.5% Fe, silicon 1.5% and total carbon 3.5%, the average analyses of the materials from which it is to be made being as follows:

Material	SiO_2	Al_2O_3	CaO	MgO
FUEL — Connellsville Coke...	5.40	2.90	.90	.69
ORE — Ashland, Gogebic....	6.34	2.70	.38	.24
FLUX — Limestone.....	1.44	2.12	46.23	6.43

(CONTINUED)

Material	Mn	P	S	Fe	C
FUEL — Connellsville Coke...		.012	.83	.....	89.5
ORE — Ashland, Gogebic	.30	.04	.009	53.49	....
FLUX — Limestone.....	...	.005	....	.....	....

It is assumed furthermore, that the ratio of the acids to the bases in the slag will be as

1 is to 1.2, a common ratio in smooth running slags.

The first step is to find the amount of flux needed for the fuel and for the ore. To do this it is necessary to find its AVAILABLE base as indicated on page 87, as only such proportion of its bases are free to act as flux for the fuel-ash and ore as are left after fluxing the acids in its own composition.

FLUX — Available base and weight of slag.

Acids		Bases	
SiO_2	1.42	CaO	46.23
Al_2O_3	2.12	MgO	6.43
	3.54		52.66

3.54 × 1.2 (bases in slag = 1.2 × acids) = 4.25 bases needed.

52.66 (bases present) — 4.25 (bases needed) = 48.41, surplus or available base.

Weight of slag formed =

$SiO_2 + Al_2O_3 + CaO + MgO$

1.42 + 2.12 + 46.23 + 6.43 = 56.20% [weight of stone.

FUEL — Flux needed, weight of slag and available carbon (page 107).

Acids		Bases	
SiO_2	5.40	CaO	.90
Al_2O_3	2.90	MgO	.69
	8.30		1.59

8.30 × 1.2 (ratio to bases acids) = 9.96 bases needed.

9.96 (bases needed) — 1.59 (bases present) = 8.37 deficiency to be made up by the limestone.

In each pound of limestone there is .484 lbs., available base, so that $\frac{8.37}{.484}$ = 17.29 lbs. limestone to flux ash.

To flux the sulphur it will require an amount proportional to their atomic weights: —

$$\underset{(56)}{CaO} : \underset{(32)}{S} :: \times : .83$$

$\times$ = 1.45 lbs. of the oxide, or,

$\frac{1.45}{.48}$ = 3.02 lbs. limestone.

17.29 + 3.02 = 20.31 lbs. stone per 100 lbs. fuel.

Weight of slag formed =

8.30 + 1.59 + .83 + (20.31 × .562) = 22.13 lbs.

The usual assumption that the slag requires 25% of its weight in carbon to melt it, is sufficiently close for practical purposes.

25% of 22.13 = 5.53 lbs. carbon.

89.5 (fixed C) — 5.53 = 83.97 available carbon.

ORE — Flux needed, weight of slag.

Acids		Bases		
SiO_2	6.34	CaO	.38	
Al_2O_3	2.70	MgO	.24	
	9.04	MnO	.13	(Assume ⅓ goes into slag as oxide.)
			.75	

Since the pig is to be 94.5% metallic iron and the ore is 53.49% iron, it is obvious that it will require $\frac{94.5}{53.49}$ or 1.767 tons of ore for one ton of pig iron.

The pig is to have 1.5% Si which is reduced from the silica in the gangue of the ore, or in other words, 1.5 lbs. silicon for each 100 lbs. of pig.

$\frac{60}{28} \times 1.5 = 3.21$ lbs. silica, SiO_2

There is, then, 3.21 lbs. of SiO_2 in the 176.7 lbs., of ore which it is unnecessary to flux, equal to 1.87% SiO_2.

9.04 — 1.87 = 7.17

7.17 × 1.2 (ratio bases to acids in slag) = 8.60 bases needed.

8.60 (bases needed) — .75 (bases present) = 7.85 deficiency to be made up by limestone.

$\frac{7.85}{.484}$ = 16.22 lbs. stone per 100 lbs. ore.

1.767 × .1622 = .2866 tons stone per ton pig.

Weight of slag formed from ore =

(.0717 + .0075) × 1.767 = .1399 tons.

Weight of slag formed from stone =

.562 × .2866 = .1611 tons.

.1399 + .1611 = .301 total slag from ore per ton of pig.

FUEL REQUIREMENTS — As noted before, practice has shown approximately 25% carbon

required for melting the slag so that .301 × .25 = .075 parts carbon for this item. To this must be added carbon for reducing and melting the pig and for its impregnation with carbon.

In the average furnace the careful calculation of the heat balance, that is the determination of all sources of heat which enter the hearth, and the accounting of the heat given off or absorbed in one form or another shows that about 66 lbs. of carbon is required for the reduction, impregnation with carbon and melting of each 100 lbs. of 1% silicon pig iron, with an extra 5 lbs. of carbon for each 1% of silicon. The amount is accurately determined only by the heat balance, and is dependent on the temperature and volume of the blast, the character of the burden, temperature and volume of the gases, the carbon ratio, temperature, weight of slag and iron, etc., but with a blast at about 1000° F. the above figure is sufficiently close for the purpose. For the reduction, impregnation and melting of 1.5% silicon pig there is needed:

.66 + .025 = .685 parts carbon.

.685 (carbon for pig) + .075 (carbon for slag) = .76 parts carbon.

With coke of 83.97 available carbon there is required:

$$\frac{.76}{.8397} \times 2240 = 2027 \text{ lbs. coke per ton of pig,}$$

or, adding 3% for braize, 2100 lbs.

The slag from the ore per ton of pig will equal .301 × 2240 = 674 lbs. and from the coke, .2231 × 2027 = 452 lbs., or a total of 1126 lbs. per ton of pig.

The total stone will be, for the ore,

.2866 × 2240 = 542 lbs., and for the coke,

.2213 × 2027 = 448 lbs., or a total of 990 lbs.

The basis of the charge is taken as the fuel, and the ore and stone proportioned accordingly as above. The amount of fuel in the "round" as it is termed, varies with the size of the furnace, — in a fairly good sized furnace being 5 gross tons. On this basis, the charge in the above proportions would be:

Coke	11,200 lbs.
Ore	22,000 lbs.
Stone	5,500 lbs.

The coke charge is usually kept constant and the ore and stone varied to suit the condition of the furnace operation as it changes from time to time.

OPERATING *the* FURNACE

CHAPTER V

BLOWING IN

WHEN a new furnace is to be blown in or an old one has been relined, the first operation is to thoroughly dry out the brickwork. Although the brick, if properly laid, are with the thinnest possible joints, yet each course is bedded in a grouting or slurry of fire clay and all interstices filled with it so that there is in reality a large amount of water to be evaporated before the furnace lining is thoroughly dry. This drying is accomplished either by wood fires built in the hearth or by gas burners inserted there. The fire should be but light at first, increasing in intensity as the furnace dries. Often ten days to two weeks is considered sufficient time in which to do this work, but two weeks should be the minimum limit. In view of the immense importance of the lining as affecting the life of the furnace, too much

stress cannot be laid upon the necessity for its thorough drying-out.

In a number of instances the development of hot spots soon after blowing in has undoubtedly been due to shrinkage cracks opening up in the mass of brickwork because of premature blowing in of a too green furnace. However, the realization that the saving of a few days at one end of the campaign may be made at cost of months on the other, has undoubtedly led to a more rational view of the situation.

It is common practice to spread on the bottom of the furnace hearth two or three feet of coke braize. This serves to prevent chilling of the first iron melted, as well as chilling of the iron notch which sometimes occurs when this precaution is omitted. Upon the braize is charged coke, well up toward the tuyeres, then a mass of kindling wood covered in turn with charcoal for a couple of feet in thickness, surmounted by a blank of coke extending at least above the mantle. This procedure is sometimes varied by the use of layers of cord wood on end, placed either at the bottom of the hearth and extending up 12 to 16 feet, or the wood may be started on a scaffold at or about the level of the tuyeres, the space beneath being filled with kindling. With the coke blank is charged a small quantity of limestone sufficient to flux the ash of the coke, as well as sometimes a

certain amount of slag to increase the volume of slag and "wash" out the hearth.

The charges of fuel, ore and flux are now begun, but with a much larger fuel ratio than when the furnace is in operation, the ratio decreasing gradually toward the top so that it is about one to one when the furnace is filled and ready to light. The burden or proportion of ore is gradually increased as the furnace is operated until in about two weeks it is carrying its normal burden.

The furnace is lighted through the tuyeres with all openings save the bleeder tightly closed, the valves at stoves and boilers also shut; a light blast is then turned on. The smoky gas which soon appears at the bleeder is ignited and allowed to burn until the smoke disappears, showing that the wood is consumed. The bleeder is then closed and the valve farthest from the furnace on the boiler line may be opened and lighted after the gas has had sufficient interval to pass through the downcomer and piping, forcing out the contained air. Every care is necessary, as explosions are very liable to occur should the gas mingled with air come in contact with a flame. At certain plants it is an invariable rule that the gas shall not be lighted until 48 hours after starting.

Soon after the furnace is lighted, the slag begins to accumulate in the hearth and a little

later, the iron as well. As soon as the slag reaches about the level of the tuyeres it is drawn off at the cinder notch and this operation repeated as often as necessary until sufficient iron has accumulated to tap. The first iron is liable to be "off" iron, i.e., it is not of the proper grade or analysis, as it is at this time impossible to have such temperature and composition of slag as to produce iron of given analysis. Within a few casts, however, the furnace is usually brought up to the production of iron of the grade desired.

CHARGING — In the modern automatic skip-filled furnace, the stock is usually dumped from the railroad cars into bins from which it is run direct to larry cars, automatically weighed, and from these dumps into the skips.

As the furnace continues in operation, the column of material constantly descends and the furnace is as constantly filled at the top by whatever may be the filling and distributing device used.

The furnace is charged by "rounds," i.e., so many pounds of fuel, ore and coke constitute a round. The fuel usually is a constant anywhere from 5,000 to 15,000 pounds in weight, dependent on the size of the furnace, while the ore and stone are varied to suit the varying requirements of output as influenced by the operation of the furnace. The fuel is usually charged first, then the ore and stone, this order

being maintained with each round. As it is most important that the height of the stock be kept fairly constant and well up to the top of the furnace, it is continually tested by rods inserted at the top or is followed by some form of automatic stock recorder.

OPERATION OF FURNACE — In general, the condition of the furnace and its operation are best told by the character of the slag and iron it is producing. The relative temperature of the hearth can also be well judged by its appearance as seen through the eye-sights at the tuyeres. The color of the hearth interior will vary from a dull red tint when the furnace is "cold," a temperature at which the outlines of the coke, etc., are easily seen, to an incandescent white heat. The appearance of the hearth as noted merely serves as a check in the determination of its temperature, but is most clearly evidenced by the character of its product. The slag gives a very good indication, the desideratum being that it be hot, fluid and grey.

As previously indicated, it is impossible to get hot iron without having a hot slag, but the slag although hot may be so difficult to fuse as to be viscous and fail to flow clean from the furnace. Furthermore, unless fluid, it fails in a measure to fully perform its function as the various impurities are by no means as readily soluble in it. Its fluidity, in fact, acts much like that of water, serving as a vehicle

of solution for any given compounds which it may hold dissolved.

A slag, on the other hand, may be so fusible as to be very fluid at comparatively low temperature and so pass through the furnace quickly, without attaining a temperature which enables it to completely perform its office. Such a scouring slag is often very destructive in its action upon the furnace lining.

Its color gives a good indication of its composition as well as of the temperature of its formation. Hot basic slags usually show a greyish color and granular fracture when cold. A few very high in lime tend to crumble and slake on continued exposure to the atmosphere. As the composition of the slag becomes more and more highly acid, a vitreous appearance is noticeable on the outer edges of the fractured piece, this vitreous condition extending deeper and deeper in accordance with the acidity of its composition. Slags from a cold furnace are usually darker in color and also vitreous. Their glassy appearance is due, in a measure, to the excess of silica, less having been reduced and incorporated in the iron as silicon. This excess of silica in the slag tends to absorb more oxide of iron, giving a darker color.

Should the furnace be running too hot and with a given burden reducing too much silicon, i.e., producing an iron too high in silicon, its temperature may be reduced by a proper

change in the burden, increasing the proportion of ore. The effect of such a change would be felt only after the new burden had reached the hearth but in spite of this delay such practice would be much preferable to the introduction of cold blast which would tend to raise the fusion zone higher in the furnace. It is the aim of the furnace manager to operate under as nearly constant conditions as possible and the quantity of wind blown into the furnace is kept practically constant as well as the temperature of the blast.

Should the hearth temperature become lower, the amount of silicon reduced becomes less and less and the sulphur increases. This condition affects also the carbon in the iron, not appreciably as regards its total quantity but as determining what proportion of the carbon shall be in the form of combined carbon and what graphitic. In this instance, with low silicon and high sulphur, the major portion of the carbon is permitted to remain in the combined form, producing mottled or white iron, an off grade.

The rise in sulphur accompanying the drop in silicon due to a cold hearth is attributable in a measure to the following causes:

1st. Decreased basicity of slag due to its greater silicon content.

2d. Excluding action of silicon upon sulphur in the molten metal.

In addition to the above effects of the cold hearth, there sometimes occurs floating on the slag or entangled with it, what is known as "buck shot." This is iron which is but partly carburized and at the temperature of the furnace has been unable to penetrate the slag and mingle with the remainder in the hearth. These shot will analyze considerably higher in sulphur than the run of the iron.

The remedy for the above condition depends on its severity. If the tendency is but slight, the putting on of a fresh stove may be sufficient. Continued coldness, however, indicates that the burden is too heavy and the quantity of ore must be diminished. The total carbon is little influenced either by the burden of the furnace or the temperature as such. It is, however, influenced indirectly by whatever affects the silicon content of the iron, because of the excluding effect of this element. Phosphorus also has the same tendency. Irons high in manganese on the other hand have an attraction for carbon and this element is consequently higher in iron with a considerable percentage of manganese. In general, the total carbon in the ordinary class of pig irons runs very closely from 3% to 4%.

BANKING THE FURNACE — For one reason or another it is sometimes necessary to curtail the production of a furnace, leaving it in condition to start on short notice. This is usually done

by charging a heavy fuel blank of coke together with enough stone to flux its ash, the amount of such coke dependent on the time which the furnace will probably be idle. This is followed by light charges of ore much below the normal amount and these, in turn, often by a blanket of fine ore serving to seal the interstices. As soon as the coke blank reaches the bottom of the furnace, the blast is taken off, the furnace tapped clear of iron and slag, the tuyeres removed and all openings bricked up and luted with clay as nearly air-tight as possible. With this procedure there is comparatively little opportunity for combustion to occur in the furnace. When the furnace is again to be put in operation it will usually be found that the stock has been somewhat reduced by the very slow combustion that has occurred with the small quantities of air unavoidably admitted. Just how much the stock is lowered depends, of course, on how well the work has been done and how long the furnace has been banked, but it is perfectly possible to close the furnace so tightly that after the first few days no settling whatever occurs. If any settling has occurred the furnace is then filled with very light charges of coke and ore, or with merely a blank of coke. The ashes are raked out through the tuyere openings, the tuyeres inserted and the blast put on. In this way the furnace will, within a few casts, be making good iron.

There are circumstances where there is no time for the preparation of banking the furnace as outlined above. In such a case nothing can be done except to tap the iron and cinder, remove the tuyeres and close all openings tightly. It is naturally a much more lengthy and difficult matter to start again under such circumstances, and it is necessary to cut through the chilled material at the iron and cinder notches before the furnace can again be put in operation. Dynamite may even be a necessity to break up the frozen material in the hearth. On putting on the blast, very light charges of ore are used with blanks of coke so that the hearth may successfully regain its lost heat.

BLOWING OUT — In blowing out the furnace for relining or such repairs as necessitate the ending of its campaign, charging is stopped and the stock allowed to settle. The top becomes very hot under such circumstances, and it is usually necessary to cool the interior of the furnace by water sprays to avoid explosions. As soon as the stock gets within a few feet of the tuyeres, the blast is taken off, tuyeres removed and the balance of the stock shoveled out, when the furnace is ready for dismantling.

OPERATION OF STOVES — Assume a typical plant of four stoves. After erection, the brickwork of the stoves should be dried out with a slow fire in much the same manner as the furnace proper and when it is desired to blow in, a

heavy wood fire is built in each at the base of the combustion chamber. This is usually succeeded by a coal fire, bringing the stoves to as high a temperature as possible by this means.

The "raw" gas from the furnace as the latter is lighted, is then turned on all four stoves as the gas is forced from the furnace by the light blast from the blowing engines which, at this period, are operating at but a fraction of their normal speed. Due to furnace conditions, this gas is naturally much higher in CO than when the furnace is operating regularly. When mixed with air the gas is highly explosive.

After the raw gas has been allowed to burn in the stoves for some time and has thus raised the temperature to say 800° or 900° F. the blast from the engine is forced through one of these stoves while the other three are still on gas.

This single stove in from one-half hour to an hour is then cut out and a fresh stove put on, the furnace thus gradually working up to normal conditions, when with a four-stove plant any given one will be in blast one hour and on gas three hours.

The more usual method is to keep the stove on for a definite time, normally an hour, regardless of the precise drop in temperature within that period, rather than to put on a fresh stove after a drop through any exact temperature range. This procedure may be

varied somewhat as made necessary by changing conditions in the furnace.

CLEANING — More or less frequent cleaning of the stove is necessitated by the accumulations of flue dust formed of minute particles of coke, ore and limestone carried over by the blast. Fragments of brickwork are also dislodged and collect in the flues, thus choking them and interfering with the blast. The amount of flue dust which may accumulate varies widely with the operation of the furnace and the character of the burden; with fine, soft ores there is naturally more dust carried over than with coarser ores of the same approximate composition, and conditions are often aggravated by hard driving. Moreover, with certain ores such as those which burden the furnace for ferro-manganese, much trouble is experienced from the accumulation of dust, although the ores in their natural condition are neither soft nor fine. In the furnace they readily disintegrate to a powdered form and are carried over by the blast, making the gas dirty. The effect of any considerable accumulation of dust in the stoves as it clings to the flue walls, is to greatly reduce the heating capacity, as accumulations offer much resistance to the taking on and giving off of heat. This may become in a short time so aggravated that the stove is practically without efficiency unless cleaned.

Another serious trouble occasioned by flue dust of certain compositions is the severe fluxing action on the brickwork at stove temperatures. This is due to the lime and other fluxes in the dust and is in some instances sufficient to lower the refractoriness of the brick several hundred degrees, completely honeycombing the top checkers where accumulation is greatest.

In a plant of any size the cleaning crew is always at work unless the gas is first passed through some form of washer or scrubber, and the stove will often require some attention at least every two or three weeks. Cleaning is accomplished by weighted brushes and scrapers let down through the flues and the accumulation is then shoveled through the cleaning doors at the bottom.

FURNACE TROUBLES — There are few situations where experience is of more value and where prompt and decisive action is more necessary than with the management of a blast furnace plant when trouble occurs.

Destruction of Lining — The trouble with a lining usually shows itself in a hot spot or the heating of the shell at various heights above the mantle. This may occur at any time from a few weeks to months, or even much longer after blowing in, and results either from a crack in the lining which allows the hot furnace gases to reach the shell or from the wearing away of

the lining in a particular spot so thin that the heat affects the shell. The usual treatment is to put on a water spray, thus keeping the plates cool, and if the spot is fairly high in the furnace the lining will sometimes build up so that the spray may ultimately be discontinued or the spot at least does not increase in area. If due to a crack in the lining, grouting is often pumped in through a hole in the shell, and with this aid the lining may then give entirely satisfactory life.

As to the causes of the trouble, there are a number of contributory conditions which may have considerable to do with it, but as the action goes on entirely hidden from view, it is naturally often difficult to point definitely to the exact cause in any given case.

There is no question whatever that faulty stock distribution has been more than any other one thing the main factor in these troubles. A case in point is that of four new furnaces successively going into blast at one plant at intervals of but a few weeks, all on the same burden and with the same brands of material for lining. After a few weeks run, each developed a hot spot at exactly the same location, about 20′ above the bosh and on the side opposite the skip. It was, of course, beyond the bounds of possibility that, had any material in the lining been defective, it should have by chance *been collected* in each instance in the same spot

and trouble occur nowhere else. A change in the method of stock distribution completely remedied the difficulty. The skips so dumped as to allow the coarser and heavier portions of the material to roll to one side and the furnace gases taking the line of least resistance, channeled up that side. The use of soft coke also accentuates any defects in the stock distributing device and in a number of instances has undoubtedly been the prime factor in causing furnace troubles. A bell that is not properly proportioned will also cause the same difficulty. The large number of tops and distributing devices which have been designed to overcome this difficulty indicate the importance attached to it by blast furnace men.

Subject as the lining is, not only to the severe abrasion of the stock as it is charged into the furnace and continues on its way downward, but to the fluxing and disintegrating effect of the molten and semi-molten mass at all points below the fusion zone, it is yet somewhat protected from the above, and from the action of the corroding gases by the carbon coating deposited on the brickwork soon after the furnace is blown in.

The majority of managers realize the importance of this phase of operation and allow sufficient time for such carbon deposit to form. *It is undoubtedly* assisted in its formation by *the running of* a hot and basic slag.

Another evil of too hastily bringing the furnace up to normal working conditions is the opening up of shrinkage cracks which may extend entirely through the wall to the furnace shell. The cause for this has already been discussed but the fact will bear emphasis. Not only should the brickwork of the furnace be well dried out before the furnace is lighted but even then a considerable period should elapse before it is brought up to maximum capacity. In certain plants, having record output and minimum costs, four weeks' time is not considered too long.

In a number of cases the lining has apparently melted and worn away at a point so high as to preclude the possibility of sufficient heat to affect fire brick of even a very mediocre grade under normal conditions, this pointing to the evidence of fluxing action.

The height at which this trouble occurs corresponds with the height at which are deposited the alkaline carbonates referred to in the succeeding chapter and experiments as to the effect of such carbonates on high grade refractory material in test kilns show that so active are these fluxes in combination with the clay as to greatly lower the point of fusion. Although not positively determined, there seems little question that this fluxing action occurs as *above outlined*.

Break-Outs — The lining may be destroyed and a break-out occur at almost any point below the melting zone, but is usually more serious when below the line of the slag or iron. A break-out in the bottom of the hearth may allow the iron to escape and find its way out under the foundations. Inasmuch as molten iron as such has little or no action in disintegrating brickwork, the opening of cracks or fissures in the hearth is due to rupture from expansion, settling foundation or defective work in laying the bottoms.

Loss of Tuyeres and Cooler Plates — The most frequent and annoying cause of trouble is the burning or melting of the tuyeres, which occurs from one of two causes, i.e., stoppage of water supply or the action of molten iron on the metal of the tuyere. In the case of stoppage of water supply, the water in the nose of the tuyere is soon turned to steam and the metal then burns away.

Normally the tuyeres are considerably above the line of the molten iron, but in case of an explosion or slip, portions of the charge are sometimes projected into the hearth, raising the iron and slag temporarily above the tuyere level. They are also subject to the dripping of the molten iron as it emerges from the fusion zone and drops through the slag to the hearth. The tuyere nose is often partially protected by accumulations from the stock,

but the mass of semi-molten material may at any time so arrange itself as to allow the dripping iron to impinge directly upon the tuyeres, in which case, if long continued, the iron forms an alloy with the bronze or copper and melts away.

A leaky tuyere is more often evident by its effect upon the product of the furnace than in any other way. The effect of the water discharged in the hearth is, of course, to cool it, and there is then encountered the dark, vitreous cinder common to cold hearths and a corresponding rise in the sulphur of the iron. It is also sometimes shown by the increase in hydrogen in the escaping gases and may first be brought to the attention of the furnace man by the back-firing of the gases in gas engines of certain types where such are installed.

The only thing to do is to replace the tuyere as soon as possible, which is done by shutting off the blast, disconnecting and jarring loose the defective tuyere and inserting a new one.

Pillaring — This condition of the stock in the hearth is essentially due to insufficient penetration of the blast, and occurs as a column of comparatively cold stock usually at the center of the hearth. The remedy is either decreasing of the tuyere area or increasing the blast to secure better penetration. It may be sometimes necessary to put in additional tuyeres.

Scaffolding — Scaffolding usually occurs either in the upper portion of the bosh wall or the extreme bottom portion of the inwall; in other words, at about the region of the mantle, and it is due to the adherence or building out of the stock partially supported by the walls of the furnace. It is primarily due to contraction in the zone of fusion which in turn may be attributed to any of the different causes producing irregular working. If the zone of fusion is, for instance, suddenly lowered, the pasty mass at its top tends to adhere to the surrounding wall, forming an incrustation projecting into the furnace. This mass offers severe obstruction to the passage of the gases and to the descending stock. The gases are driven to one side and may even cut out and channel the furnace wall on the side opposite the scaffold. Anything which will temporarily raise the zone of fusion may soften the obstruction, when, if the engines are slackened, the obstruction may often be loosened by the weight of the burden above.

The trouble may become so severe as to require more stringent methods, such as the insertion of an extra tuyere or even the use of dynamite. Continued trouble from this source is usually due to faulty design of the furnace, too heavy burden, too much flux or any of the *numerous* factors which tend to irregular working.

When the incrustation falls away, it often brings with it a mass of brickwork, leaving a hole where the stock tends to lodge and increasing the chances of further hanging, particularly as the freshly fractured brick is protected by no carbon coating.

Wedging — This term is often used to describe the condition of the stock in the upper portion of the furnace when it becomes so tightly wedged or bound for the time being as to prevent its falling. It is due to carbon deposition and naturally occurs with those ores which deposit the larger amount of carbon in their reduction. It is more particularly prevalent with the Mesabi ores and does not occur with the Magnetites, however fine may be their state of sub-division. In some cases the deposit is much more than the original volume of ore, so that the tendency to wedging and corresponding obstruction to the descent of the stock is easily explained. As the effect of the deposited carbon is to fill the voids in the stock, the mass becomes exceedingly dense and the gas can with difficulty penetrate it. The pressure of the blast usually rises rapidly, the stock beneath meanwhile settling until at length the wedged stock falls. This is usually termed a slip and is accompanied by more or less of a so-called explosion and a certain amount of stock is often thrown from the furnace, while the shock may be so severe as to entirely raise the

furnace top. There is much discussion as to the true underlying cause of this apparent explosion. By some it is thought due to the sudden release of the pressure of the trapped blast while others believe a true explosion occurs. If the latter be the correct assumption it may possibly be due to the very finely divided state of the carbon which, upon access to oxygen, may explode at the temperature involved in a manner not dissimilar to a so-called "dust" explosion.

The prevention of wedging obviously lies primarily in a proper design of the lines of the furnace and in carrying such a burden as does not require too large a proportion of ore depositing excessive carbon. The use of increased quantities of limestone with such ores is said to mitigate the condition through the solvent action of the CO_2 gas given off as the ore is heated. It has, however, the disadvantage of correspondingly increasing the fuel consumption.

Chilling — The chilling of the hearth naturally results from whatever may cause an insufficient supply of heat at that place and may result from insufficient fuel supply, leaky tuyeres or the falling into the hearth of large masses of comparatively cold stock, due to a severe slip.

The remedy depends on the severity of the chill. If due to too little fuel, the cooling of the

hearth is gradual and may again be brought to the proper temperature by the charging of coke blanks. If merely local, as for instance, a hard tapping hole due to leakage of a tuyere or plate near the front of the furnace, the tapping hole may be opened with an oxyhydrogen blow pipe, oil burner or electric arc.

If the whole hearth is chilled, as may be the case where a mass of stock is precipitated into it, every opening may be closed due to the displacement of the metal and slag, and the only way is to cut out the cinder notch and possibly a couple of tuyeres to provide an opening for the blast and an escape for the products of fusion, gradually working the hearth back to its normal condition with the aid of the hot blast from the two tuyeres, and with others, as fast as they can be brought into commission.

FURNACE REACTIONS

CHAPTER VI

GENERAL DISCUSSION

ALTHOUGH the results of the reactions which occur in the chemistry of the blast furnace may be very simply expressed and in few equations, yet the exact steps involved in arriving at these results and the precise order of their precedence are matters which experiment and investigation of the future will undoubtedly considerably develop. The occurrence or non-occurrence of certain reactions in given zones is so much dependent on the exact temperature of that zone, and so interrelated are the many reactions, a number of them occurring simultaneously at various phases of the process, that the simple equations by which the results are expressed are sometimes misleading as to the complexity of the reactions going on within the furnace.

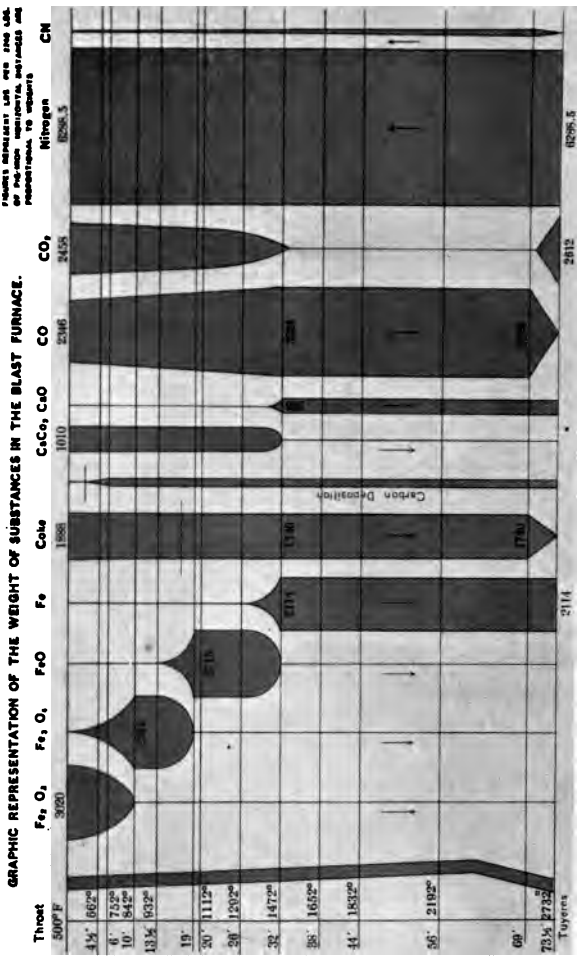
We have to deal only with the three members of the charge entering at the top — the ore,

flux and fuel, and the air blown in through the tuyeres at the bottom. This air may be at the temperature of the atmosphere, cold blast, so-called, although now seldom used, and on no large furnaces; or the hot blast at a temperature varying from 800° to 1000° F. or even 1300° or 1400° F. although the latter is seldom regularly maintained in practice in this country. This hot air blown through the tuyeres at a pressure of from 5 lbs. to 15 lbs. per square inch meets the glowing coke before the tuyeres, and the oxygen of the air immediately unites with the carbon of the coke and forms carbonic acid gas, CO_2. This gas, however, is at once changed to the monoxide on coming in contact with the surrounding particles of incandescent coke, i.e., $CO_2 + C = 2\ CO$. Whatever water may be in the air as vapor is decomposed in contact with the incandescent carbon, the oxygen uniting with the carbon, leaving free hydrogen. This series of reactions, which is nothing more or less than that of the gas producer, is very simply expressed as follows:

1st. Carbon burned to carbon dioxide.
$C + 2\ O = CO_2$

2d. Reduction of carbon dioxide by hot coke.
$CO_2 + C = 2\ CO$

3d. Incandescent carbon decomposing water and combining with oxygen.
$C + H_2O = CO + H_2$



FROM CAMPBELL

The comparatively inert nitrogen of the air remains as a whole unchanged and merely dilutes the other gases, although this is not strictly true, as will be later considered. This then, is the composition of the heated volume of gases rising to mingle with the descending charge of ore, fuel and flux — the composition, that is, except as modified by changes in and the effect of reactions upon members of the charge itself. By volume, just above the tuyeres it is about 37% carbon monoxide, CO, with the balance nitrogen, save for perhaps 1% hydrogen and hydrocarbons. So important is the temperature in determining the various reactions occurring during the descent of the charge that these are shown in the accompanying diagram, page 146, as indicated by Campbell for what may be called a standard type of 90′ furnace. The temperatures and their relative depths are indicated, but it should be borne in mind that these are by no means absolute, but vary with the height of the furnace, its lines, character of ores and burden, and the rate at which the furnace is driven. Moreover, the temperature shown at the tuyeres as 2732° F. in general is conceded to be about 3500° F. much nearer the theoretical temperature at that point.

FLUX — Considering in detail the changes occurring in the various constituents of the charge, the flux will be the first considered.

As noted before, this may be either a comparatively pure limestone, $CaCO_3$, or a

magnesian limestone more or less closely approaching the composition of a true dolomite. As the stone is charged into the furnace, it is at first entirely unaffected by the top temperature, say 500° F. save for the gradual absorption of heat. Neither is it affected by any chemical action of the gases encountered. As it descends, however, acquiring constantly increasing temperature, it comes to a point where, due to this heat, its dissociation begins according to the equation: — $CaCO_3 = CaO + CO_2$.

Under normal atmospheric pressure, this change begins to occur at about 1050° F. although it does not become complete much under 1500° F. the temperature of 1050° F. corresponding to a depth in the furnace of approximately 20′. However, as the gases in the furnace have added to the normal atmospheric pressure, that of the blast at an average of possibly 8 to 10 pounds, so the temperature must be correspondingly increased, and under actual blast furnace conditions, its dissociation probably begins at about 1100° F. but is uncompleted until practically 1600° F. is reached. These temperatures correspond with a depth of about 20′ and 45′ respectively, so that from the time the limestone reaches a depth of 20′, CO_2 gas begins to be released, the action becoming livelier as the temperature rises and is completed at a depth of 45′, leaving the oxide CaO to continue its descent, steadily

absorbing more heat until it reaches the zone of fusion well above the tuyeres. Here it unites with the silicious portion of the gangue and the ash of the coke, and the whole softens and melts, dropping as slag or cinder to the hearth.

The carbonic acid gas, CO_2, which has been liberated during the descent of the stone, has mingled and risen with the volume of heated gases rushing upward through the shaft of the furnace, and has been by no means inactive. The action is identical with that occurring in the case of CO_2 gas formed by the combustion of the atmospheric oxygen and the carbon of the coke, as the blast enters the furnace, i.e., that of the so-called "carbon transfer" or the breaking down of the CO_2 in contact with the incandescent coke into CO.

$$CO_2 + C = 2\ CO$$

This occurs under furnace conditions, at all temperatures over approximately 750° F. or from 15′ to 20′ from the top of the furnace, while, as we have seen, the zone through which the carbon dioxide is given off from the stone, has its beginning about 20′ from the top and extends to a depth of possibly 45′ or more. Thus, throughout this whole distance, or until the gases are cooled below 750° F. the hot coke is in this way robbed of just so much of its carbon, an abstraction of heat units for which no return is made. It is an absolute loss

so far as the economy of operation of the furnace is concerned, but passing off through the downcomer, its energy is again available in the burning of the gases in stoves, in the gas engine or under the boilers.

FUEL — The coke is the least affected of any of the members of the charge in its descent. Save for the comparatively small amounts of carbon abstracted by the CO_2 generated by the decomposition of the stone, the coke arrives almost at the level of the tuyeres in its original condition except for the absorption of heat. Here it is burned, the carbon reduced to CO rising with the ascending column of gas and its ash unites with the flux and is incorporated in the slag.

ORE — The reduction of the ore, usually an oxide of iron, is effected by both carbonic oxide, CO, and by carbon. The exact order in which the various reactions resulting in this reduction occur, and their thoroughness, depend both upon the temperature and upon the composition of the gases in contact with the ore.

In general, most of the change occurs in the upper part of the furnace and begins at temperatures considerably lower than those at which either coke or limestone are affected. With the more easily reducible ores, deoxidation by CO gas begins between 400° F. and *500° F.* but is much more active at from 800°

to 900° F. The reduction due to carbon does not begin until a temperature of about 750° F. is reached.

As the ore descends in the furnace and reduction becomes more nearly complete, it is left in the form of very finely divided metallic iron or iron sponge, impregnated with more or less carbon deposited during the reduction. Upon arriving at the zone of fusion, slightly above the tuyeres, this iron sponge becomes plastic and finally melts, absorbing a portion of the impregnating carbon, and drops through the accumulated slag to the hearth.

The reactions which occur in the process, as briefly outlined, are simple ones, but the exact order of their occurrence and the degree to which each is completed at any given stage, are by no means easy to determine.

The following equations represent the action of carbonic oxide on the oxides of iron encountered:

(a) $3\ Fe_2O_3 + CO = 2\ Fe_3O_4 + CO_2$

(b) $Fe_3O_4 + CO = 3\ FeO + CO_2$

(c) $FeO + CO = Fe + CO_2$

The results of the above reactions may be expressed as follows:

$$Fe_2O_3 + 3\ CO = Fe_2 + 3\ CO_2$$

Unfortunately, reduction is actually completed by no means as simply as this, because by a change of temperature the product of

equations (b) and (c), i.e., FeO and Fe, may in turn be oxidized by the carbon dioxide liberated in their formation. Such reoxidation may be expressed by the following equations:

(d) $2\ FeO + CO_2 = Fe_2O_3 + CO$
(e) $2\ Fe + 3\ CO_2 = Fe_2O_3 + 3\ CO$

The reactions of CO_2 upon the metallic sponge and the lower oxides begin, according to Campbell, at 570° F. although more active at higher temperatures and varying with the relative amounts of CO_2 and CO in the gases present. Since CO_2 is a necessary product in the equation by which the iron ore is reduced, it is evident that the reduction can never be wholly completed by the action of CO alone, as the CO_2 evolved will, at high temperatures, reoxidize to some extent the metallic iron. The excess of CO, however, whose presence is due to the carbon transfer allows this reduction to be much more nearly complete than would otherwise be possible.

For the completion of this reduction, we must look to the reducing effect of carbon which is deposited throughout the mass of the ore as it is partially reduced and descends in the furnace. There is much discussion in accounting for the precise equation involved in its deposition and the manner of its occurrence. There is still much to be learned. It is claimed by some that it is due to the power of the metallic iron or

iron sponge and the lower oxides to split up the CO gas in accordance with the equation:

$$2\ CO = CO_2 + C$$

In this, the iron does not itself form part of the reaction, but acts as a calalytic agent, as does the well known platinum sponge in a number of familiar instances in the laboratory.

Others have claimed that deposition can occur only by the oxidation of metallic iron according to the following equations:

$$Fe + CO = FeO + C$$
$$2\ Fe + CO_2 = 2\ FeO + C$$

However, deposition begins at about 420° F. and at that temperature metallic iron has not yet been reduced from the ore except by a reaction which itself produces carbon dioxide. Neither does CO_2 react upon metallic iron until a temperature of over 500° F. Although either of the above equations involving oxidation by carbon monoxide or dioxide may account for most of the deposition occurring at high temperatures, the only reaction which can occur at the temperature at which deposited carbon is first actually found is:

$$2\ Fe_2O_3 + 8\ CO = 7\ CO_2 + 4\ Fe + C$$

Whatever the reactions involved, its deposition becomes more active as the temperature is raised, but ceases at about 1000° F. and it is probably this carbon which removes the last traces of oxygen, completing the

work of the CO. The reactions between the oxides and carbon are expressed as follows:

$Fe_2O_3 + 3\ C = 2\ Fe + 3\ CO$
$Fe_3O_4 + 4\ C = 3\ Fe + 4\ CO$
$FeO + C = Fe + CO$

The matter of the deposition of carbon is a most important one, both on account of the reducing effect of the carbon itself and the effect its deposition has in disintegrating the ores. Its reducing power is evident at all temperatures above about 720° F. The quantity evolved in the reduction of various ores is apparently much dependent upon the mineralogical classification of the ores, Laudig finding the following maximum and minimum:

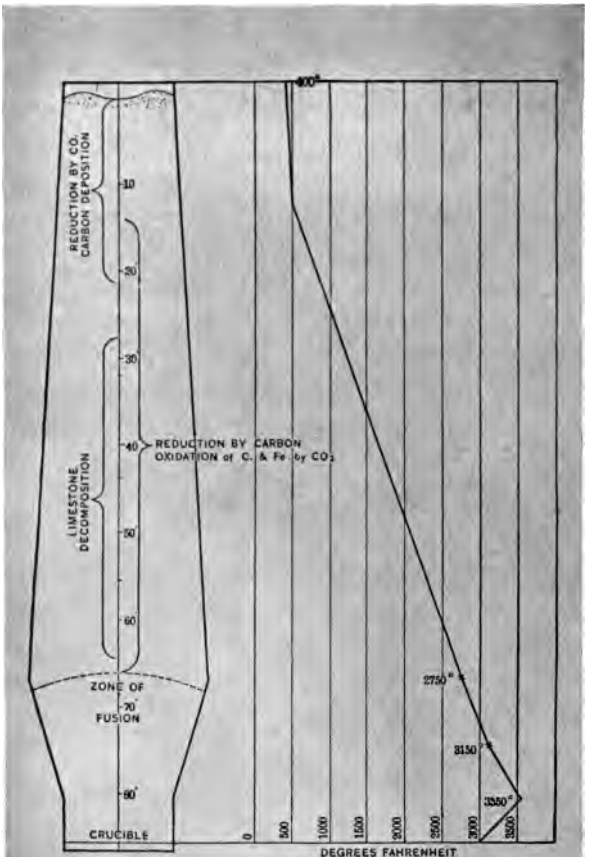
	Maximum	Minimum
Mesabi Ores.....	36.40	10.20
Brown Hematites..	24.92	.98
Magnetites.....	Practically nil.	

The above figures are the weights of carbon deposited in per cent. weight of the ore, and indicate that the mass of the ore is thus sometimes increased two or three times the original volume. There is no question but that the porosity of the ore has an important bearing on its ease of reduction, possibly due to the fact that the carbon may thus be more easily deposited throughout its mass and effect its work of reduction. It may also partially explain the fact of the greater difficulty usually encountered in the reduction of the Magnetites.

The value of the more easily reducible ores, such as those of the Mesabi range, is seen from the fact that the greater part of this reduction and the consequent liberation of CO_2 gas takes place in the upper zone of the furnace and at a temperature at which the gas does not react upon the coke, CO_2 passing out of the furnace without being reduced to CO, with the consequent expenditure of heat and higher fuel consumption due to it. Conversely, the higher fuel consumption with Magnetite ores, is also undoubtedly explained by the higher temperature and consequent greater depth at which its reduction occurs.

The carbon deposit is subject to the solvent action of CO_2 at temperatures of about 760° F. and above, so that a very considerable portion of it may be thus dissolved during its journey through the furnace although the deposition itself ceases, as has been mentioned, at about 1000° F. Were it not for this fact, there would undoubtedly occur great trouble in scaffolding and clogging of the descending mass, due to its great increase in volume; such trouble is in fact by no means uncommon with the very fine and easily reducible ores.

As a summary of the changes occurring in the blast furnace, the graphic representation shown on page 146 may again be referred to.



Temperature Gradient of Blast Furnace.

FROM FORSYTHE

This figure shows not only the general order of occurrence but the degree to which each reaction is completed under the usual conditions existing at any given level in the furnace.

There are differences of opinion, however, among a number of authorities as to some of the points as indicated on this diagram particularly as regards the progress of reduction of the ore. In accordance with Bell's experiments it apparently often occurs, as a whole, at higher levels than as shown in this diagram. Limestone, too, was by no means always entirely decomposed at 32′ from the top, but at times persisted well below the upper bosh line.

Deposited carbon is usually assumed to be more nearly absorbed by CO_2 than as indicated and the temperature at the tuyeres or just above is, in a majority of cases, considerably higher than here shown.

As showing the temperatures usually considered more representative of average modern practice, the accompanying diagram showing the temperature gradient, as given in Forsythe's work, will be of interest.

CYANIDES — Although the greater portion of the nitrogen of the air entering the furnace as blast may be considered as inert, yet a certain part unites with the incandescent carbon of the coke, forming cyanogen, Cn, which in turn

combines with whatever sodium and potassium may be present, forming cyanides. Owing to the affinity of the cyanides for oxygen with the formation of cyanates, their action, whatever it may be quantitatively, is actively reducing in its nature. The cyanates are volatile and join the rising volume of gases until decomposed by carbon dioxide, leaving the carbonate as deposited in the upper portions of the furnace. This carbonate may be carried down by the descending charge and again combine with cyanogen near the tuyeres. There is thus an ever increasing quantity of cyanides present and to them is attributed by some the improvement in the running during the period following the blowing in of the furnace.

CARBON RATIO — This term is used to express the ratio of the CO in the escaping gases to the CO_2. The ratio $\frac{CO}{CO_2}$ has a direct relation to fuel consumption inasmuch as the greater the proportion of CO_2 the greater the heat developed in the furnace, and the less heat units escape as CO. It means, however, that the greater the dilution of CO, the reducing gas, the less the reducing power in the gases. The ratio is often brought below 1.5, however, with good results from the standpoint of reduction.

The following table from Forsythe gives representative analyses of these gases and is of

interest as showing their variation at the several levels:

Depth Feet	Per cent. by Volume CO.	CO_2.
Top	29.5	11.0
4	29.5	10.5
8	27.0	8.0
10	32.0	7.0
12	33.0	7.0
14	31.0	6.5
16½	34.1	2.2
20	35.1	0.7
39	35.0	1.1
52	35.2	1.5
65	35.9	0.5
70½	36.6	0.0
Tuyeres	37.7	0.8

DRY-AIR BLAST — In considering this phase of modern blast furnace practice, it should be realized that it is in no sense a mere theoretical refinement, but aims to regulate the one big variable in the operation of the blast furnace. All the other materials, the ore, stone and coke vary at the most but a few per cent. in their composition from time to time, while the moisture in the air may vary more than 100% from day to day, and of this air the production of a ton of iron requires almost double the weight of the other raw materials.

As Gayley points out, 1 grain of moisture per cubic foot of air means practically a gallon of water per hour for each thousand cubic feet of air per minute, blown into the furnace. In other words, for a furnace blowing 35,000 feet of air per minute, not a large furnace, there would be blown in 35 gallons of water per hour for each grain of water per cubic foot in the atmosphere.

The amount of this vapor in the atmosphere varies with the temperature and humidity, the average the year through for Pennsylvania, for instance, being about 55° F. and a humidity of 70%. At 55° F. the atmosphere, if saturated, would have 4.92 grains of moisture per cubic foot; at 70% humidity, 3.44 grains per cubic foot.

In the so-called dry blast a large portion of the moisture in the atmosphere is removed by refrigeration previous to the introduction of the air into the stoves. Any moisture on entering the furnace at the temperature encountered at the tuyeres is immediately dissociated by the intense heat into the component gases:— $H_2O = 2\ H + O$. In doing this, it absorbs the same amount of heat as is generated in burning hydrogen to make an equivalent quantity of water. In other words, in accordance with the atomic weights of the elements, if 18 parts of water by weight are dissociated, they will make 2 parts by weight of hydrogen

and 16 parts by weight of oxygen, absorbing the same heat as generated by burning 2 parts by weight of hydrogen. Two pounds of hydrogen on burning to the condition of steam, give off 103,500 B.T.U., or this is the heat absorbed by the dissociation of 18 parts of steam, or for each pound, 5,750 B.T.U.

However, on the credit side should be placed the heat gained by the burning of the oxygen evolved to CO, C + O = CO. This heat equivalent equals,

$$\frac{12}{18} \times 4450 \text{ B.T.U.} = 2965 \text{ B.T.U's per lb. steam.}$$

$$5750 - 2965 = 2785 \text{ B.T.U.}$$

This is the net loss by absorption per pound of vapor in the blast.

The amount of air necessary to burn the fuel for smelting 100 pounds of pig varies with the temperature of the blast, but a fair average may be taken at 5300 cubic feet. With the amount of moisture contained under average conditions, as previously assumed for Pennsylvania (3.44 grains) the total moisture will then be:

$$\frac{5300 \times 3.44}{7000 \text{ (grains per lb.)}} = 2.6 \text{ lbs.}$$

$$2780 \text{ B.T.U} \times 2.6 = 7235 \text{ B.T.U.}$$

The above figure represents the heat lost per hundred pounds of iron smelted, due to the moisture in the atmosphere or, expressed in

coke consumption, somewhat over 50 pounds of coke per ton of pig made.

The Gayley process, as installed at several plants, has removed by cooling to 25° or 30° F. approximately 65 to 70 pounds of water per ton of pig smelted during some of the more humid months of the year, a theoretical saving in the consumption of coke of some 55 pounds per ton of iron, but the actual saving has proven far greater.

A brief summary of the results as covered in Gayley's paper on the Application of Dry-Air Blast to the Manufacture of Pig Iron, shows the following:

	Natural blast, Aug. 1–11, inclusive.	Dry blast, Aug. 25–Sept. 9, inclusive.
Average moisture	5.66 grs. per cu. ft.	1.75 grs. per cu. ft.
Average daily output	358 tons	447 tons
Average fuel consumption	2,147 pounds	1,726 pounds
Cubic feet air	40,000 per min.	34,000 per min.
Temperature of blast	720 degrees F.	870 degrees F.
CO in gases	22.3 per cent.	19.9 per cent.
CO_2 in gases	13.0 per cent.	16.0 per cent.
Temperature of gases	538 degrees F.	376 degrees F.
Flue dust	5.0 per cent.	1.0 per cent.

From this it will be seen that the actual saving far exceeds any that can theoretically be accounted for either by the mere elimination of moisture or rise in temperature of the blast.

There has been much discussion regarding it, but probably the greatest saving is in reality attributable to the securing of uniformly favorable operating conditions. Regularity is a prime essential for economical running and with these conditions uniformly good there is no need for the excess fuel necessary to provide for contingencies.

It would not, however, be the part of wisdom to assume that such enormous saving could be made at all plants and under all conditions, but it is probably safe to say that the average plant can decrease its coke consumption at least 12% and increase the production 10%, while in very many cases it is perfectly feasible to raise these percentages to 14% and 12% respectively. This, of course, refers to what may be relied upon the year through and not merely for short periods under the stress of record breaking output.

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